MINING engineering

JULY 1953

CORNWALL MINE - CONTINUOUS OPERATION SINCE 1742

MINING engineering

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COVER

Candle and electric hat lamp are reminiscent of contrast at Cornwall Banks, where modern mining methods prove their value against historical setting dating back to Colonial America. More about the modern era starting on page 670.

FEATURES

ARTICLES

TRANSACTIONS

—Personnel Service—

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

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Management Executive, 36, married. Non-metallics industry, broad experience in personnel relations, quality production, costs, safety, quarrying and processing; some underground mining and metallics experience. Prefers western U.S.A. Available July first. M-30-523-E-1-San Francisco.

Mining Engineer, employed, 35, married. Sixteen years background technical training and practical experience in exploration, examination, planning, development and operation in mechanized coal mining; some experience with ore mining and two years executive business training to offer a reliable company needing a person with top managerial training. Will consider production or sales anywhere. M-31.

POSITIONS OPEN

Geologist, young, for laboratory and field work for mining company. Salary, $4200-$4800 a year. House furnished; three year contract. Excellent climatic conditions. Location, Cyprus. Y8739(c).

Resident Engineer, graduate, with several years experience in coal mining, including underground surveying and mapping, and development planning, for large western U.S. coal mine. Y8646.

Mining Engineer with experience in exploration and development for mining of hard rock. Must have experience in underground surveying. Give experience, age, and salary requirements. Location, eastern United States. Y8462.

Mining Engineer, young, graduate, with some experience in surveying and mapping. Should be able to assume responsibility and get along well with associates. Duties will include mine and surface surveying and mapping, calculating, and performing other related engineering work. Salary to start, $4200 a year with opportunities of advancement and salary increase. Location, South. Y8371.

Sales Engineers, with mining or construction background, to handle sales of mechanical underground and surface loaders. For underground equipment should have knowledge of underground mining conditions; and for surface loaders, should be familiar with heavy construction work and open-pit mining. Salary open. Location, New York, N. Y. Y8342(b).

Technical Sales Engineer for export department, to assist in technical sales, be familiar with overseas operation and able to handle French and German. Should have mining engineering background and be familiar with heavy underground and construction machinery. Salary open. Location, New York, N. Y. Y8342(d).

Engineers. (a) Mill Superintendent experienced in wet gravity concentration and with some knowledge of roasting of concentrates. Single status for six months. Must speak Spanish. Salary about $7800 a year. (b) Assistant Mine Superintendent with Latin American background, preferably mining experience in vein mines in Mexico, Chile or other South American countries. Single status; graduate not necessary. Should be competent in mine operation and production. Salary $6000 a year. (c) Mining Engineer with some knowledge of Spanish. Capable of doing necessary surveying underground and on the surface. Will have to bring maps up to date and keep them posted; do sampling where required and map principal features of underground geology. Ability to do correct mapping and neat drafting essential. Single status. Salary, $6000 a year. Location, Bolivia, S. A. F8587.

Junior Mining Engineer, recent graduate or one years experience, for survey and flotation in ores and general engineering. Any special courses in ore dressing desirable. Salary, $4200 a year. Location, North Carolina. Y8544.

Geologist with experience on gold and silver, to estimate potentiality of such a property in Mexico. Prefer resident of Mexico. Headquarters, New York, N. Y. Y8551.

Mining Engineer to superintend pegmatite mining operations. Should have five to ten years operating experience, at least half of it in open cut mining. Experience in feldspar, mica or other forms of pegmatite mining particularly desirable. Salary open. Location, New England. Y8561.

Engineers. (a) General Superintendent, under 50, with underground iron ore mining experience. Salary, $10,000 to $12,000 a year. (b) Mine Foreman, young, to take charge of shift operations. Location, eastern United States. Y8260.

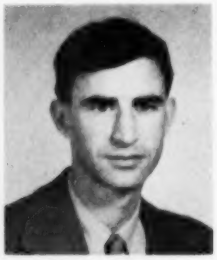
Assayer with nonferrous, secondary smelting experience. Salary, $4160 to $5200 a year. Location, northern New Jersey. Y8261.

Mining Engineer, E.M., must have had four or more years experience in quarrying hard rock, crushing and servicing, and possibly some underground experience. Knowledge of Zeology. Will be in charge of general engineering having to do with quarrying, crushing, and screening of rock and rock products. Company manufactures building materials. Salary up to $500 per month. Company will negotiate fee. Location, Chicago, Ill. R9630(b).

Engineer, young, with mining or cement mill experience, to sell machinery for export. Salary, $4500 to $5000 a year. Location, New York, N. Y. Y8755.

Geologists experienced in underground water resources. One for twelve months work in Turkey; the other for over fifteen months work in Southwest Africa. Salaries open. Y8122.

Meet The Authors

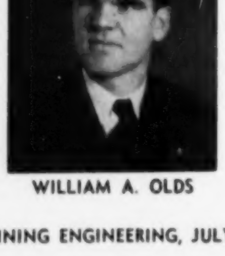


JOHN T. REED

John T. Reed (P. 705) has been with University of California for more than a year as a lecturer and research engineer. Prior to that he spent six years with Braden Copper Co. in Chile as a shift boss, level foreman, assistant mine foreman, and field hydraulic engineer in the electrical department. Mr. Reed received his Bachelor of Science in Mining in 1944 from the University of California and later earned a Master of Science in Mining.

John F. Knudsen (P. 689) received a Bachelor of Arts in Education and Physical Sciences from Arizona State College and did graduate work at the University of Wyoming and the University of Southern California. At school he was a member of Pi Delta Sigma, Lambda Delta Lambda, Blue Key, and graduated with Honorable Distinction. He currently is with the Industrial Hygiene Department, Kennecott Copper Corp., Garfield, Utah. Mr. Knudsen taught physical sciences in a high school before going into the mining field.

William A. Olds (P. 696), of Titanium Metals Corp. of America, Henderson, Nev., worked his way through the University of Washington as a bus boy, laboratory assistant, and research assistant on an Army project. He still found time to play chess, bridge, and take part in church activities. Soon after graduation he joined the U. S. Bureau of Mines; his major duties were in connection with design and investigation of a heavy media pilot plant. Mr. Olds is an active member of the American Legion and various church groups. In addition he follows the baseball and football world with interest, and is a photographer and chess player.



WILLIAM A. OLDS

Ross C. Anderson (P. 711) was born in Miami, Oklahoma and attended the University of Kansas. Before going to work for Ozark-Mahoning Co., he was employed for seven months as an architectural draftsman in Kansas City, Mo. Mr. Anderson now lives in Tulsa, Okla. Golf, tennis, and swimming compete with gardening for his spare time, but he says his favorite is the last named hobby.

W. F. Boericke (P. 679) is currently with Hayden Stone & Co., New York City, and is a resident of Long Island. He was employed by New Jersey Zinc Co. for 11 years; was assistant to H. Foster Bain, Institute Secretary, from 1925 to 1931; chief valuation engineer for four years with the Securities Exchange Commission; and chief evaluation engineer with the Bureau of Mines, seven years. Mr. Boericke has authored many Transactions, in addition to articles which appeared in several leading magazines. His main relaxation is bridge.

L. C. Raymond (P. 679) has been engaged in mine examination and evaluation for Ford, Bacon, & Davis Inc., of New York City. He is their senior mining engineer. He has been associated with several new domestic mining projects as well as several large foreign developments in South America, the West Indies, and Canada. Eight years of his early professional career were spent in exploration and examination work through the western states and in Canada and in mine operation with the Mountain Copper Co., Ltd., as junior engineer, geologist, and assistant superintendent. Mr. Raymond also served four years with the U. S. Tariff Commission as mineral specialist preparing economic studies for Congress dealing with foreign controls and trade restrictions. A graduate of Oregon State School of Mines and MIT, Mr. Raymond says he likes to paint, handle explorer scouts, and fish when time allows. His three children are his main interest.

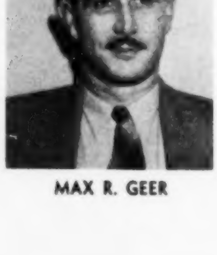
William I. Weisman (P. 711) is a member of the American Chemical Society, Engineers Club of Tulsa, and vice-chairman, Resources Div., Research Department, Tulsa Chamber of Commerce. It seems that his various activities and his position with Ozark-Mahoning Co. leave little free time. Mr. Weisman has been with Ozark-Mahoning since 1948. Previously, he was employed by the War Assets Administration, from 1946 to 1948. Mr. Weisman was a Navy ordnance officer during World War II, and is a graduate of the University of Minnesota.

BERNARD YORK

Bernard York (P. 705) calls his scores at his favorite hobbies—trout fishing, bird hunting, and prospecting, just fair. Mr. York is currently an associate professor of mining, Div. of Mineral Technology, University of California. He gained experience in mining activities from 1932 to 1940 while employed in Nevada and California. In 1940 he joined the faculty of the University of California. He holds a Bachelor of Science in Mining and an Engineer of Mines degree, from the University of Nevada.

H. F. Yancey (P. 696) earned his Ph.D. after attending the University of Missouri and the University of Illinois. Mr. Yancey headed the Solid Fuels Mission to Germany for the Joint Chiefs of Staff and served with General Headquarters of SCAP in Japan in 1950. He won the Percy Nicholls Award of the AIME Coal Div. in 1952. When not busy at his job with the U. S. Bureau of Mines in Seattle, he tries his hand at steelhead fishing in Washington and rainbow trout angling in British Columbia.

Max R. Geer (P. 696) is chief, Coal Branch, Region II, U. S. Bureau of Mines, Seattle, Wash. A native of Ohio, he attended Ohio State University and the University of Washington, earning a Bachelor of Mining Engineering, Master of Science, and Master of Engineering. Mr. Geer has been with the Bureau of Mines since 1935. He is currently involved in coal preparation research. Ten of the 27 papers he has written on coal preparation have been published by the AIME. Woodworking, fishing and skiing are his spare-time interests.



MAX R. GEER

"Bedeviled Copper" helps to build a battleship

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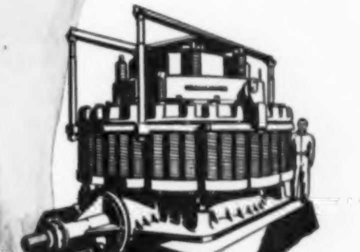

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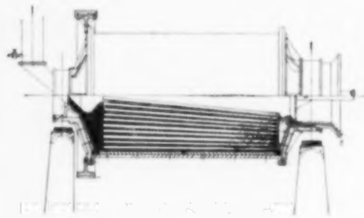

The Marcy principle of low pulp line, fast discharge of mill content results in lower overall costs than when dry crushing is used... the discharge is more uniform, a higher percentage of discharge is finished product. Thus the load is lightened or capacity increased, on the final stage of grinding.

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Books for Engineers

ORDER YOUR BOOKS THROUGH AIME—Address Irene K. Sharp, Book Department. Ten per cent discount given whenever possible. Order Government publications direct from the agency concerned.

Non-Ferrous Ore Dressing in the U. S. A. Report of the Technical Assistance Mission No. 54. *Columbia University Press.* $2.50, 209 pp.—This is a report prepared by a Technical Assistance Mission of European experts visiting the U. S. studying different aspects of the industry and comparing them with practices in their own countries.

Bibliography on Uranium in Arizona, Nevada, and New Mexico, by Margaret Cooper. *Geological Society of America.* 38 pp., $0.25., 1953.— Offers bibliography and index of published literature on uranium and thorium and radioactive occurrences in the U. S.

Advances in Geophysics, by H. E. Landsburg and others. *Academic Press,* 362 pp., 1952.—Deals with a wide range of geophysical thought covered by eight authors. Statistics, meteorological information, exploratory work and other matters are covered.

Engineering Valuation and Depreciation, by Anson Marston, Robley Winfrey, and Jean C. Hemstead. *McGraw Hill Book Co.,* $8.00, 508 pp.—This is a major revision of Marston and Agg's *Engineering Valuation* dealing in concepts and procedures involved in the appraisal of industrial, commercial, utility and natural resource properties.

The Basis of Mine Surveying, by M. H. Haddock. *Chapman & Hall, Ltd.* $4.26, 301 pp., 1952.— Provides a groundwork in the subject and deals fully with mathematic requirements. Mechanical methods of computation are presented.

Chemical Processing of Wood, by Alfred J. Stamm and Elwin E. Harris. *Chemical Publishing Co., Inc.* $12.00, 595 pp., 1953.—A comprehensive presentation of information on the chemical utilization of woods, especially of wood residues and inferior species of wood, to produce modified wood products, pulp products, and various derived chemicals.

Geology and Mineral Deposits of Pre-Cambrian Rocks of the Van Horn Area, Texas, by Philip B. King and Peter T. Flawn. *University of Texas,* $5.75 cloth bound and $5.00 paper bound, 218 pp., 1953.—Bureau of Economic Geology of the University of Texas and the U. S. Geological Survey cooperated in preparing the text.

Lime and Limestone, Part I, by N. V. S. Knibbs and B. J. Gee. *H. L. Hall Corp., Ltd.,* 113 pp., 1953.—Discusses the origin, occurrence, properties, chemistry, analysis, and testing of limestone, dolomite and their products, and the theory of lime-burning and hydration.

The Colorimetric Determination of Copper with 2,2-Diquinolyl in Minerals and Ores, by R. J. Guest. *Canadian Department of Mines and Technical Surveys.* $0.25, 18 pp., 1953.—A colorimetric procedure for the determination of copper is described. According to the report, the simplicity and rapidity of the method lend it to routine work.

The Determination of Aluminum by the Fluorophotometric Method, by J. B. Zimmerman, *Canadian Department of Mines and Technical Surveys.* $0.25, 12 pp., 1953.—A method is described by which a wide range of aluminum concentrations may be determined speedily and with approximately ±5 pct accuracy for samples containing less than 1.0 pct aluminum.

(Continued on page 648)

The Mining Journal Annual Review. *The Mining Journal Ltd.* More than 250 pages. $1.00, 1953.—A summary of events throughout the mining world during 1952. Technical and economic aspects are covered in a country by country review.

Directory of Indian Mines and Metals, compiled by P. K. Ghosh. *Mining, Geological and Metallurgical Institute of India.* 208 pp., 1952. —The text was compiled to present a general picture of the Indian mineral industry. Prices quoted are those current during February 1952. The directory supplies a list of firms, and presents an introduction to Indian minerals in respect to application, treatment, distribution, and commercial aspects.

Handy Technical Dictionary. *Central Book Co.* $20.00, 1088 pp., 1953.—Technical terms in English, German, French, Italian, Portuguese, Spanish, Polish, and Russian are arranged in language groups with each group arranged alphabetically. Subjects dealt with comprise light and heavy machinery, parts components and sections, electrical plants, and trade and domestic tools and appliances.

U. S. Geological Survey Publications

Circ 158 Industrial Clays of the Columbia Basin (Circulars are issued free of charge by the U.S.G.S.).

Circ 202 Thorium mineral occurrences in Alaska.

Circ 207 Reconnaissance for radioactive deposits in Cook Inlet region, Alaska.

Circ 213 Report on Jo. Reynolds area, Clear Creek Cty., Colo.

Circ 219 Reconnaissance of various uranium and copper deposits in Western U.S.

Circ 220 Selected papers on uranium deposits in the U.S.

Circ 225 Geochemical association of columbium and titanium.

U.S.G.S. circulars are issued free of charge.

Recent U. S. Patents

2,628,787 Apparatus for analyzing particle size distribution. (2/17/53) *Sharples Corp.*

2,628,717 Sulphide flotation reagent. (2/17/53) *American Cyanamid Co.*

2,628,827 Air-mechanical flotation machine. (2/17/53) *Mining Process & Patent Co.*

2,629,493/494 Iron ore flotation with carbohydrate xanthate. (2/24/53) *Earl H. Brown.*

2,629,650 Removal of iron impurities from phosphate rock. (2/24/53) *International Minerals & Chemical Corp.*

2,630,207 Underground mine conveyor. (3/3/53) *Joy Mfg. Co.*

2,630,226 Miner's gold pan with trap. (3/3/53) *Frank Streng.*

2,630,306 Subterranean retorting of oil shale through tunnels. (3/3/53) *Socony-Vacuum Oil Co.*

2,630,307 Recovery of oil from shale in place. (3/3/53) *Carbonic Products Co.*

2,630,308 Coal mining and loading machine. (3/3/53) *Jeffrey Mfg. Co.*

2,630,309 Centrifugal reverberatory furnace (3/3/53) *Frederick C. Ramsing.*

2,630,369 Hydrometallurgical process for treating vanadium and uranium ores (3/3/53) *Climax Uranium Co.*

2,630,371 Process for manufacture of magnesium. (3/3/53) *Merck & Co.*

2,630,377 Continuous leaching apparatus. (3/3/53) *Henry E. Lewis.*

2,631,726 Hydraulic classifier. (3/17/53) *George A. Auer.*

2,633,089 Mine car design. (3/31/53) *Henry F. Flowers.*

2,633,240 Coal flotation process. (3/31/53) *Hercules Powder Co.*

2,633,241 Flotation removal of iron impurities from feldspar ore. (3/31/53) *Mason K. Banks.*

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FOR BLOCK CAVING: Hewitt-Robins sectionalized ore mine conveyors maintain a steady flow of ore in haulage drift or sub-drift under grizzly level . . . eliminate down time for timber repairs and replacement . . . assure less chance of breaking in stopes or arching in the chutes . . . provide faster withdrawal of block being mined.

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Manufacturers News

New Products • *FILL OUT THE POSTCARD FOR MORE INFORMATION* • Equipment

Deep Hole Drilling

Gardner-Denver Co. has announced what is said to be the first percussion drill specifically designed for deep hole drilling. The new model SFH99 permits a 4-ft change in a 7-ft drift and other features facilitate drilling

ring patterns. Shank rod location in the chuck prevents loss of rod string on down holes, and an extra heavy centralizer supports the string on up holes. The SFH99 is equipped for reverse rotation to permit one-wrench uncoupling of rods, and also has the G-D Ring Seal Shank to deliver either air or water to the rod string at high pressure. **Circle No. 1**

Welding Unit

Compact new Unitow welding unit moves direct to job under its own power. An *International Harvester* engine drives the 300 to 400 amp generator and the tractor mount. Double duty as a tractor is another feature of this unit built by *Hensel & Green Co.* **Circle No. 2**

Pump

A new magnetically driven combined pump and motor built by *Chempump Corp.* requires no mechanical seals or stuffing boxes. Suited to handling corrosive, radioactive and highly volatile liquids, the rotor is isolated. **Circle No. 3**

Air Regulator

Capable of handling large volume of compressed air with minimum pressure drop, *De Vilbiss Co.* has marketed the type HAA regulator with capacity of 80 cfm at 100 psi, capable of regulating up to 135 psi line pressure. **Circle No. 4**

Exhaust Fans

Corrosion resistant exhaust ventilators for use in industrial plants and laboratories are now produced by *American Agile Corp.* The ventilators, fabricated throughout with unplasticized polyvinyl chloride, are available in four sizes from 250 to 1670 cfm. **Circle No. 5**

Automatic Strainer

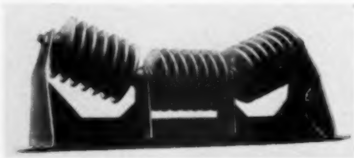
An automatic self-cleaning strainer produced by *S. P. Kinney Engineers Inc.*, for pipelines from 2 to 36 in. diam removes fine suspended particles from raw or process water and disposes of them by means of a rotating drum with straining elements. **Circle No. 6**

Hoist

Zip-Lift Electric Hoist with rope control, priced less than $200, is announced by *Harnischfeger Corp.* The new Zip-Lift comes in two models of 500 and 1000 lb lifting capacity with hoisting rates of 25 and 13 fpm. Both models are available with 12-ft and 18-ft lift. **Circle No. 7**

Conveyor Idlers

Three new lines of idlers have been announced by *Pioneer Engineering Works Inc.* The Mesabi type impact idlers are equipped with rubber segments to absorb shock of falling rock,



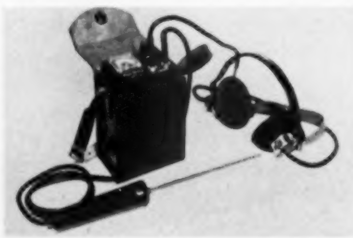
the self-cleaning return idlers are equipped with wide-spaced rubber roll segments, and the self-aligning troughing idlers incorporate a pivoted base and guide rollers. **Circle No. 8**

Electrode Holder

Cooler, lighter, and less tendency to roll are claims for an arc welding electrode holder produced by Welding Products Division of *A. O. Smith Corp.* **Circle No. 9**

Maintenance Instrument

Anco Instrument Div. has a portable electronic instrument, Elec-Dectec model V with head phones



and a milliammeter to enable the operator to "see as well as hear" noise sources in all types of mechanical equipment. **Circle No. 10**

Mobile Weigher

A new device made by *W. C. Dillon & Co. Inc.*, Van Nuys, Calif., is a combination swivel arm and scale that weighs merchandise while it is being transferred. **Circle No. 11.**

Portable Electricity

Independent source of electricity beyond power limits is provided by a gasoline driven generator introduced by *Porter-Cable Machine Co.* and available in two models: a one-man dc unit weighing 76 lb, and a two-man 135-lb ac unit. **Circle No. 12**

Bag Tires

New experimental Rolligon vehicle built for Army Transportation Corps travels on three giant *Goodyear* molded "tireless tubes." Each Roll-

igon is positioned by an axle, and driven by rotating rollers. No springs are needed. The bag-like tires will conform to rocky terrain, and provide flotation. **Circle No. 13**

Wire Rope

Using the abilities of a coil spring to flex and to resist crushing, *Jones & Laughlin Steel Corp.* has made a new wire rope with a coiled steel

spring for its core. The void inside the spring provides a reservoir for lubrication. **Circle No. 14**

Moisture Seal

Das-Seal, sold by *Dasco Chemical Co.*, is a new compound to be sprayed or painted over rocks in mine roofs for sealing moisture and seeping water. Claimed to keep mines dryer, and reduce rock fall, the sealer may be applied to damp as well as dry surfaces. **Circle No. 15**

Hot Conveying

B. F. Goodrich Co. has a new conveyor belt with a patented nylon cord breaker paired with a glass fabric belt carcass. Used successfully handling hot materials up to 500°F, the nylon cords reinforce the rubber cover and resist formation of transverse cracks. **Circle No. 16**

Rotary Drill

With the model B-52, truck-mounted, rotary drill, *Mobile Drilling Inc.* enters the deep rotary drill field with a 500-ft unit. Dual cylinder feed provides 5, 6, or 7-ft stroke and 12,000-lb pressure for auger, rotary or percussion drilling. Claimed to be low priced, the drill fits most heavy duty truck chassis. **Circle No. 17**

Free Literature

(21) CONSULTING SERVICES: *The Association of Consulting Chemists and Chemical Engineers Inc.* has issued the 14th edition of "Consulting Services," a 144-page directory of each member's qualifications and activities. A new section lists unusual laboratory equipment owned by members. A first copy is free, further requests will be honored at $1.00 per copy.

(22) MAINTENANCE: "Proper Maintenance of Control" tells what to look for in servicing control equipment. The 8-page manual from *Allis-Chalmers Mfg. Co.* has a trouble-shooting chart and lists 26 possible check points to assure satisfactory control performance.

(23) LOW TEMPERATURES: *Arthur D. Little Inc.* has made available reprints of three staff articles: "Application of Extreme Low Temperature to the Field of Metallurgy," "—Field of Electronics" and "—Field of Chemistry."

(24) PNEUMATIC CONVEYORS: A bulletin with eleven drawings and a simple text published by *Convair* explains pneumatic conveying systems used by various industries.

(25) TESTING APPARATUS: *Soiltest Inc.*, Chicago, has published a 72-page catalog describing and illustrating over 800 items of apparatus for engineering tests of soil, concrete, and bituminous materials. A special section is devoted to apparatus for chemical tests of soils.

(26) TURBINE PUMPS: A new bulletin from *Worthington Corp.* makes liberal use of photographs to illustrate wide industrial application of vertical turbine pumps.

(27) PIPE: *Babcock & Wilcox Co.* has issued a helpful guide in obtaining most economical service from pipe and tubing installations. The bulletin outlines available technical data accumulated over many years from performance reports and laboratory research.

(28) SAFETY POSTERS: Both humorous and serious approaches are used in the 756 posters illustrated in the 1953 Directory of Occupational Safety Posters published by the *National Safety Council.* Included are many posters for mining, chemical, and petroleum industries.

(29) HOSE CARE: Called "Tips on Hose Care," a bulletin from *Hewitt-Robins Inc.* tells what to do and *what not to do* in use and storage of rubber hose.

(30) PLATE MAGNETS: *Magnetic Engineering & Mfg. Co.* has a new catalog illustrating Memco Alnico-Permanent Non-Electric plate magnets made in standard widths from 4 to 48 in. and in four strengths, designed for installation on spouts, chutes, tables, or over conveyor belts.

(31) RUBBER DATA: Chemical resistance of various types of rubber is given in a check list issued by Republic Rubber Div. of *Lee Rubber and Tire Corp.* Five rubber compositions are rated for chemical resistance to 40 different materials.

(32) CARBIDE TOOLS: Nine styles of carbide tipped mining tools are featured in a new mining tool catalog released by *Vascoloy-Ramet Corp.* Auger drill bits, chain cutter bits, roof drills, and finger bits are included in the line.

(33) MINING PRODUCTS: "Products for the Mining Industry" is a Sunday supplement-type rotogravure brochure issued by *Boston Woven Hose & Rubber Co.* Photographs and captions tell the application story of hose, conveyor belt, and other industrial rubber products.

(34) TRUCK CRANE: A new 6-page illustrated bulletin on the model T-35 6-ton, ⅜-yd truck crane from *Schield Bantam Co.* contains detailed mechanical specifications, operating data, and information on truck mountings and interchangeable attachments.

(35) CASTINGS: *American Brake Shoe Co.* has a new 12-page booklet on typical industrial castings of Ductalloy (Brake Shoe's trade name for ductile iron).

(36) DITCHERS: An illustrated catalog from *Gar Wood Industries* shows the many types and sizes of Buckeye Ditchers available, their digging speeds, digging depths and widths. Data on engines, horsepower, machine weights, and ground bearing pressure are also included.

(37) STEAM TRAPS: Combining seven devices in one single unit: steam trap, air-by-pass, check valve, strainer, sight glass, temperature in-

MAIL THIS CARD

for more information on items described in Manufacturers News and for bulletins and catalogs listed in the Free Literature section.

Mining Engineering **29 West 39th St.** **New York 18, N. Y.**

Not good after October 15, 1953 — if mailed in U. S. or Canada

Please send me { More Information ☐ / Free Literature ☐ / Price Data ☐ } on items circled.

1	2	3	4	5	6	7	8	9	10
11	12	13	14	15	16	17	18	19	20
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Students are requested to write direct to the manufacturer.

Name______________ Title______

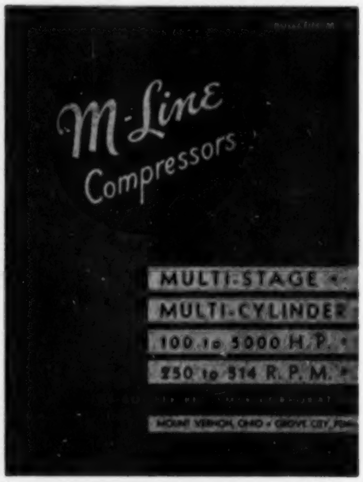
Company______________

Street______________

City and Zone______________ State______

dicator, and temperature control, the Universal Velan Trap is described in two catalogs issued by *Velan Engineering Ltd.*

(38) AIR COMPRESSOR: Descriptive data and engineering specifications on the Cooper-Bessemer M-



Line of compressors are presented in *Cooper-Bessemer's* 44-page bulletin.

(39) TRAILER PLANT: Mobile power plants no larger than a house trailer are designed by *Fischer & Associates Engineering Organization*, Cleveland. Serving as stand-by units for municipalities, they can also be used by the mining industry as well as other large installations requiring large amounts of electricity not available from usual high-line sources.

(40) NEW MOTOR STANDARDS: *General Electric Co.* provides complete information on new NEMA motor standards for 1 to 30 hp ac motors in a bulletin comparing old standards with new frame dimensions.

(41) TEMPERATURE CONTROLS: *Burling Instrument Co.* has a condensed catalog showing essential features of the 17 electrically operated temperature controls in chart form. Included is a separate description of the company's pneumatic and valve type controls.

(42) CONVEYOR CARRIERS: Simplex belt conveyor carriers with welded steel frames, and unique spun end rolls are covered in a new bulletin from *Stephens-Adamson Mfg. Co.*

(43) DISC BRAKES: Some of the varied industrial applications of the Goodyear airplane-type disc brakes are illustrated in a new brochure from *Goodyear Tire & Rubber Co.* Photographs show their use on mine shuttle cars, locomotives and loading machines, as well as on such equipment as fans and high speed saws.

(44) COMMUNICATIONS: Selective paging of supervisors and maintenance men is easily handled by IBM's Portable Paging Unit. These units, which can be plugged into ac power outlets, are described in a folder from *International Business Machines Corp.*

(45) BOLT HOLE CLEANER: *Mine Safety Appliances Co's* MSA bolt hole cleaner, tested and approved by the U. S. Bureau of Mines, is available in electrically and mechanically driven models. Further details on the dust-collecting unit are contained in a recent MSA bulletin.

(46) TORCHES: Catalog No. 100 from *The Turner Brass Works*, Sycamore, Ill. introduces a new line of L-P (Liquefied Petroleum) fire pots and torches.

(47) DIESELS: Five Cummins Diesels manufactured by *Cummins Engine Co. Inc.* and 17 standard models of Euclid haulage equipment they power are described in a 16-page booklet published by *Euclid Road Machinery Co.*

(48) STRIPPING: Thirteen photographs of Caterpillar equipment at work in strip-mining operations illustrate "Strip Mining," a new 8-page booklet published by *Caterpillar Tractor Co.*

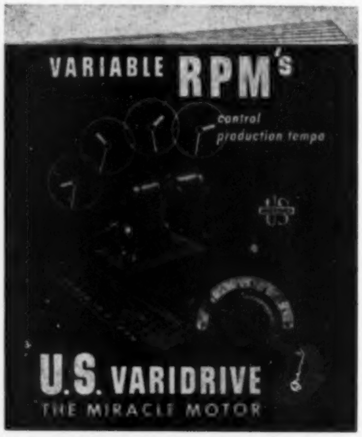
(49) VACUUM PUMP: Claiming for the first time a rotary pump that ends vapor condensation problems, *National Research Corp.* has a 16-page bulletin explaining the Gas Ballast principle and the construction and operation of the NRC Rotary Gas Ballast Pump.

(50) CORROSION CONTROL: The uses and advantages of Hagafilm, a film-forming amine for controlling corrosion in steam condensate systems, are set forth in a bulletin from the *Hagan Corp.*

(51) MATERIALS HANDLING: Information on a combination carton and drum clamp adaptable to *Elwell-Parker Electric Co.* fork trucks is now available.

(52) RUBBERIZED ABRASIVES: Adaptability, versatility, application, specifications, and prices of *Cratex Mfg. Co.'s* Rubberized Abrasives are contained in a new 8-page catalog.

(53) RPM CONTROL: A completely revised and amplified bulletin has been issued by *U. S. Electrical Motors Inc.*, manufacturers of Vari-



drive motors. It includes 80 colored illustrations of motors, mechanical features, and remote controls.

(54) CONVEYORS: Standard tubular frame conveyors are described in a bulletin from *E. F. Marsh Engineering Co.* A tubular truss is claimed to have greater strength, greater rigidity than other standard structural shapes.

(55) PIPE LINE FILTERS: A 4-page bulletin from *Dollinger Corp.* describes four new low pressure pipeline filters recommended for removal of dirt, scale, oil and water vapor from compressed air lines.

(56) PORTABLE SAW: *Ingersoll-Rand Co.* has available a folder on a new air-powered circular saw for cuts to 4⅜ in. Designed for 90 psi air supply, the S-12 air saw uses IR's Multi-Vane air motor in a saw offering maximum safety of operation.

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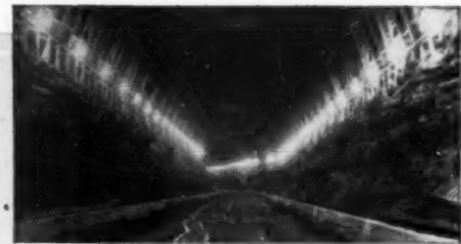
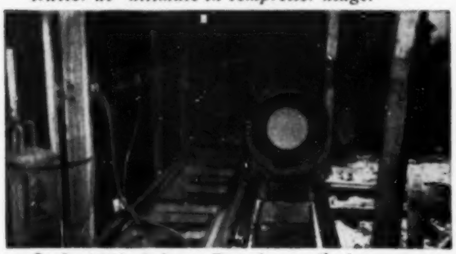


Full heading after both top heading and bench have been excavated. Excavated dia.—51 ft.

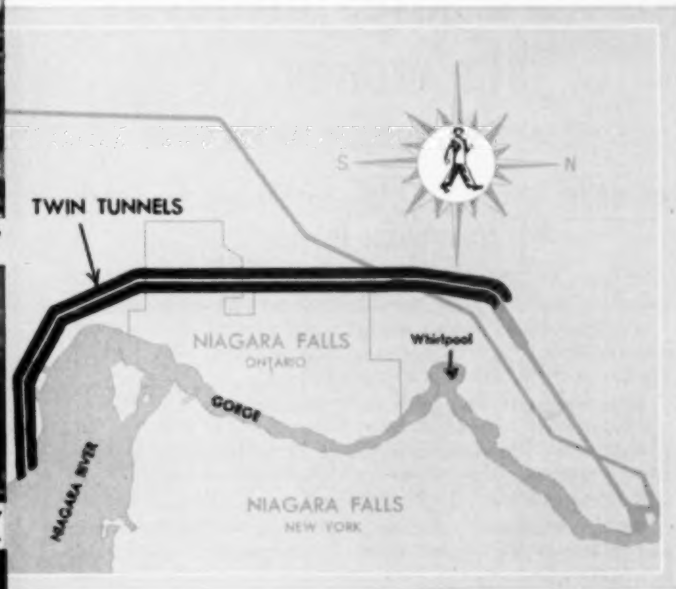
Main Jumbo for driving top heading—mounts 17 T-350 long-feed drifters on Joy Hydro Drill Jibs.

Jumbo for bench drilling—suspended from roof supports—mounts 13 Joy T-350 long feed drifters.

8 Joy WN-102 compressors—referred to by contractor as "ultimate in compressor usage."



Series 1000 Axivane Fans in ventilation arrangement. Tracks permit adding fans as desired.



PROGRESS REPORT ON THE WORLD'S LARGEST TUNNEL

SIZE: **Twin Tunnels 51' Dia., 5½ mi. length.**

LOCATION: **Niagara Falls, Ontario.**

EQUIPMENT USED **(All Joy Installation):**

114 T-350 Drifters on LW-6A Feeds on Hydro Drill Jibs.
24 WN-102 Semi-Portable Air Compressors.
Series 1000 Axivane Fans—added as needed to maintain air volume.

The world's largest tunnel is being bored out of *solid rock* with the help of Joy rock-drilling equipment, ventilating fans and air compressors.

4,375,000 cu. yds. (9,923,000 tons) of rock will be excavated to form the huge twin tunnels. These 5½ mile tunnels will convey water from the upper Niagara River beneath the city of Niagara Falls, Ontario, to a point near the Whirlpool Rapids, well below the Falls. Water thus diverted for hydro-electric power purposes would supply the water needs of 200 million people!

This is the first large tunnel operation where drilling has been mechanized. The "All Joy Equipment" installation confirms Joy's position as leader in the heavy construction and tunneling equipment market. ***Joy Manufacturing Company, Oliver Building, Pittsburgh 22, Pa.*** In Canada: *Joy Manufacturing Company (Canada) Limited, Galt, Ontario.*

Consult a Joy Engineer

JOY

WORLD'S LARGEST MANUFACTURER OF UNDERGROUND MINING EQUIPMENT

W&D C-4587

Geological Survey topographic mapping parties started initial operations in the 1953 Alaskan mapping program. One area to be covered is 17,000 sq miles in the Kantishna River, Melozitna, Tanana, Fairbanks, Healy and Mt. McKinley section. The other area to be covered is the Kenai, Tyonek, Blying Sound and Seldovia areas.

Average starting salary earned in 1952 by Columbia University School of Engineering graduates was $4000 yearly. This year's graduates can expect to start at about $4300, according to a survey taken by Columbia. Figures cover only those with Bachelor degrees. Graduates with a Master's started at an average of $500 per year more.

One of Ontario's biggest staking rushes was set off by reports of a uranium find. Prospectors and miners are reported to have swamped the mine recorder's office at Sault Ste. Marie getting claims recorded, while others grabbed off area maps before applying for mining licenses.

Seven more purchasing contracts have been signed with American and Mexican manganese producers, under a three-year project to encourage Mexican manganese production. The new contracts call for delivery of 136,000 tons of manganese to the U. S. from below the border. About 135,000 tons remain to be purchased under the program, which calls for a total of 550,000 tons.

Utah Construction Co.'s subsidiary, Marcona Mining Co., shipped some 60,000 tons of Peruvian iron ore to the Fairless Works of U. S. Steel Corp. First shipments started last April. The Marcona mine has not reached its full production potential.

The Richard mine of the Colorado Fuel & Iron Corp., Wharton, N. J. won the statewide Interplant Safety Contest for the fourth time in five years. The Richard mine had the lowest number of lost-time accidents per man-hours worked. The New Jersey Department of Labor and Industry sponsored the contest with every active mine in the state entered.

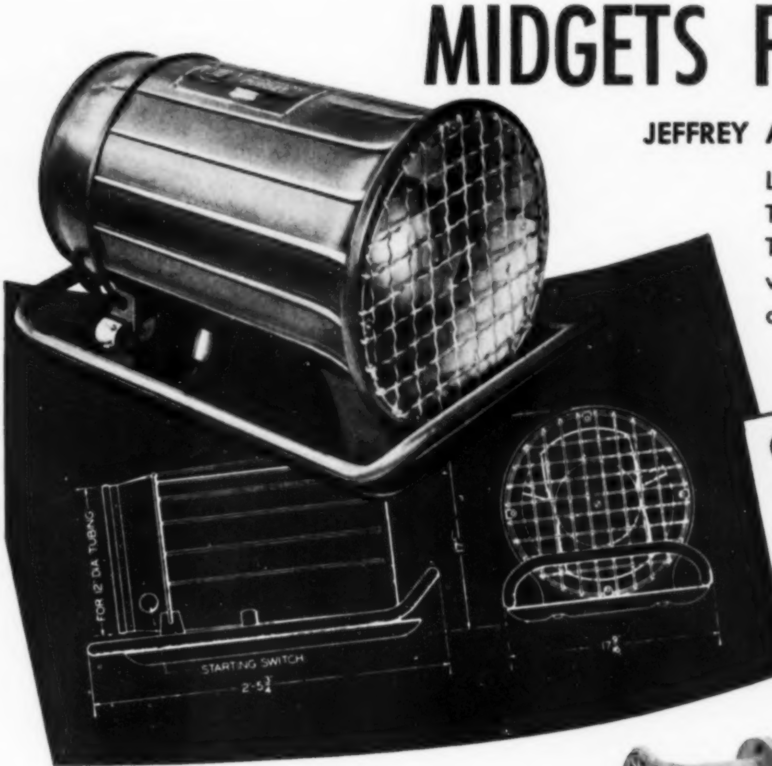
A new industry has been started in Montana by the Finlen & Sheridan Mining Co., with mining and milling of high grade barite near Elk Creek, Missoula County. Finished product is reported finding a ready market in the Midwest, Rocky Mountain states, the South, and the West Coast.

United States Steel Corp. announced that it will close its National No. 1 mine because of exhaustion of the coal body. The mine was opened in 1901 and operated by National Mining Co., U. S. Steel subsidiary. It merged with H. C. Frick Coke Co. during the 30's.

MIDGETS FOR MINES

JEFFREY AERODYNE MIDGET FANS

Low height . . . Streamlined . . . Totally Enclosed and Permissible Types . . . Furnished complete with starters for standard power characteristics.



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A.C. FAN
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D.C. FAN
5 H.P. — 6200 C.F.M.
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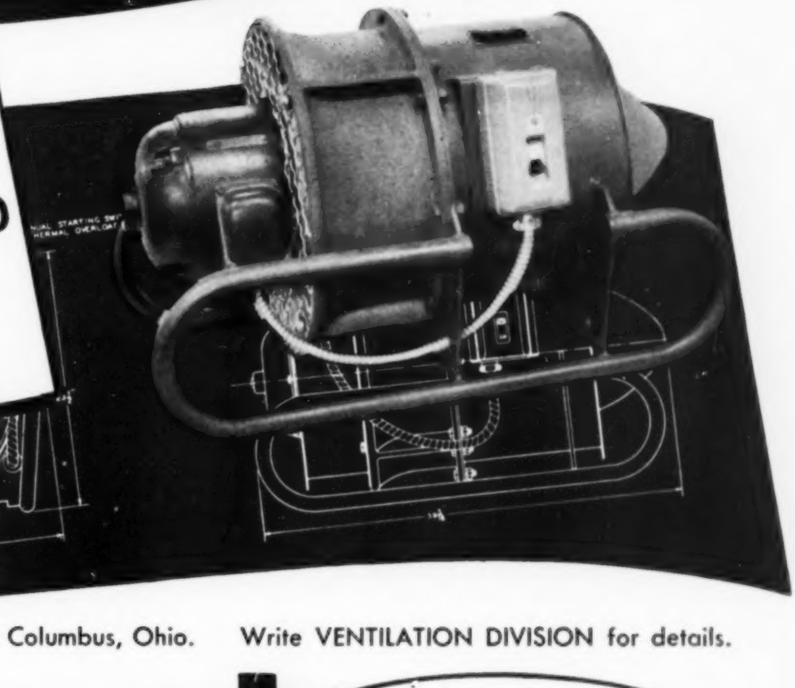

Both Fans as priced, F.O.B. Columbus, Ohio. Write VENTILATION DIVISION for details.

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National Steel Shuts Weirton Coal Operation

National Steel Corp. announced that its Weirton coal mine is to be closed for an indefinite period after June 27. The mine, near Morgantown, W. Va., was called one of the most modernly equipped mines in the country when it was opened January 1950.

No reason for closing the mine was given beyond "extremely difficult operating conditions." Some sources, however, stated that geological conditions proved difficult to solve.

Estimated reserves of the mine when it was first opened ranged to 40 years. The conveyor belt installed in the mine was termed "the world's largest continuous belt conveyor." Coal was transported through a two-mile tunnel cut into a mountain between the Monongahela River loading docks and the pit-head.

Production, some months after the mine was opened, was reported to be 1500 tons, about one third of the reported planned capacity. Annual output was originally fixed at 1.5 million tons. About 500 men were to be employed when the pit reached maximum output.

National used Weirton coal for its steel making operations. The company's most important mine is at Isabella, Pa. National also owns one third interest in Mathies Coal Co., owned jointly by National, Youngstown Sheet & Tube Co., Steel Co. of Canada Ltd., and Pittsburgh Consolidation Coal Co.

Steep Rock Expects Hogarth Output Soon

Steep Rock Mines expects production to start at its new Hogarth mine by August, with a possible 500,000 tons of iron ore shipped during the current season. This would be in addition to the expected 1 million tons from the Errington mine.

A good part of the north end of the Hogarth pit has already been exposed and shows excellent grade. It appears that a large percentage of this orebody will produce lump ore, bringing a premium of more than $1.00 per ton. Water, once a handicap at Hogarth, prior to the mammoth task of draining Steep Rock Lake, is no longer significant.

Kennecott to Start New Open Pit in Nev.

Kennecott Copper Corp. will start development work in the near future on a new open pit copper mine near Ely, Nev., according to Frank R. Milliken, vice president in charge of mining operations.

Full production is expected in 1954 from the mine to be developed from an orebody known as the Veteran, located near Kennecott's present Nevada operations. The low grade orebody, averaging less than 1 pct copper, is 1400 ft long and 600 ft wide. The deposit was mined by underground methods many years ago, although it has not been in operation since 1914.

The Veteran development is the second undertaken recently by Kennecott in this area. The first undertaking involved the moving of an entire town in order to operate new shafts of the Deep Ruth mine, which will be in full operation in 1954. Production from Deep Ruth is expected to reach about 8000 tons of ore per day.

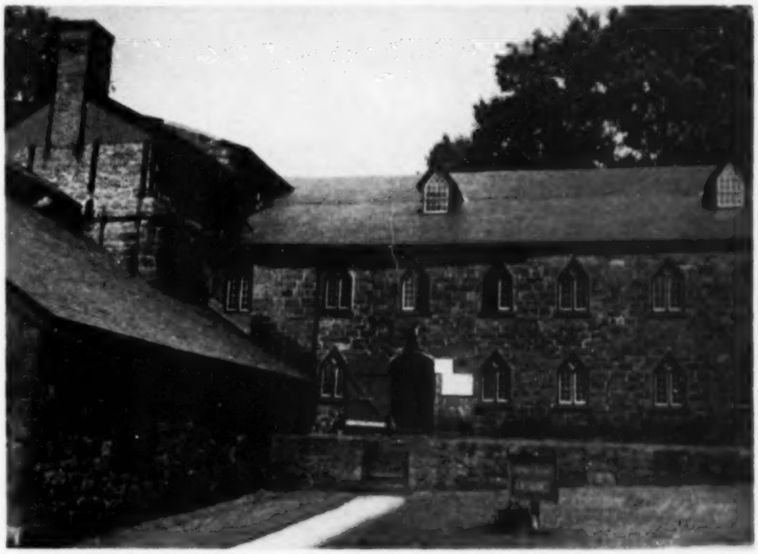
Dutch Firm May Aid Ungava Project

Fenimore Iron Mines Ltd. is reported to be negotiating with a West German steel mill and a Dutch ore and shipping firm for the development of Fenimore iron ore properties in the Ungava Bay area of Canada.

Joseph A. Retty, Fenimore president, disclosed that the N. V. Handels-En Transport-Maatschappij (Vulcaan) interests of Rotterdam will be appointed sole selling agent for the ore in Europe. Vulcaan is also said to have indicated that it will attempt to interest West German steel mills in the Quebec development. It is hoped that the German firms will supply machinery against repayment in ore.

The concession covers some 228 sq miles, extending from a point 16 miles north of Leaf Lake on Ungava Bay, southeastward about 112 miles in the Labrador Iron Ore Trough.

Mr. Retty estimates deposits contain at least 556 million tons of ores which can be concentrated to 55 pct iron and sold along the Atlantic Seaboard and in Europe. Development is $68 million.



The furnace house at the Cornwall mine was built in 1742 and is still in a fine state of preservation. Today, it is a museum open to the public and visited by thousands of tourists every year. The mine furnished iron for Washington's army and has been an active contributor in every national emergency since then.

Historic Cornwall Mine Produces Variety of Ore

Latest report of the U. S. Bureau of Mines shows that the old Cornwall mine in Pennsylvania produced a varied assortment of minerals during 1950, in addition to its main product, iron ore.

Cornwall produced 1764 oz of gold, 10,563 oz of silver, and an undisclosed amount of copper. Iron ore output was placed at 1,762,540 tons during 1950. The State Planning Board of the Pennsylvania Department of Commerce says this historic mine was the only producer of commercial cobalt in the U. S.

This month's MINING ENGINEERING contains an article covering Cornwall operations.

Crushing Plant Prepares Garfield Slag For Railroad Ballast

A modern slag crushing and screening unit has been installed at the edge of the slag dump at American Smelting & Refining Co.'s Garfield Smelter to facilitate treatment of slag, found extremely useful as ballast on western railroad grades.

Slag has become popular as ballast because of its excellent drainage characteristics which lead to longer tie life. It also provides a more rigid roadbed than other materials. It has led to a brand new activity at Garfield. Some 30 million tons of slag are already in the dump, with monthly additions of 50,000 tons adding up to what appears to be an inexhaustible supply.

The crushing and screening unit was installed by Utah Sand and Gravel Co., independent contractors. Asarco granted additional right-of-way for the slag treatment plant and necessary service spurs to the Western Pacific Railroad. Western Pacific is the present ultimate purchaser of the slag which is used for an extensive modernization program. The slag is employed for ballast on the main line.

Jaw and cone crushers are used to reduce slag to usable size. The two-step procedure minimizes fines production, a byproduct with only a limited use. Screens remove undersize and fines from the finished product.

Dumping of the molten slag takes place around a major part of the dump's roughly circular periphery. The treatment plant had to be located where future dumping would not take place. A means for removing the solid slag from the interior of the dump had to be provided without interfering with present haulage or undermining existing trackage and other facilities.

Slag is partially broken up with a modern scarifier and bulldozed to a loading point where a shuttle feeder places it on a conveyor belt. The conveyor transports it under the slag haulage track to the crushing and screening plant. A jaw crusher and a cone crusher give the desired size reduction.

Ballast is loaded on gondola cars for use by Western Pacific Railroad in its extensive modernization program. Rights of way were granted to Western Pacific by Asarco to facilitate slag transportation.

The two-step process for size reduction minimizes the production of fines, a limited-use byproduct. Slag is water-treated before crushing to eliminate dusting during preparation and loading. Vibrating screens remove undersize from the crusher feeds as well as fines from the finished product. The prepared ballast is $-1\frac{1}{2}$ $+\frac{3}{8}$ in. in size. The $-\frac{3}{8}$ size is being stockpiled at present, but there is hope that someday it may be used for road surfacing and concrete aggregate.

The finished product is lowered to the loading level by means of a *rock ladder*. The rock ladder is an enclosed tower with projecting shelves alternating on opposite sides, preventing material falling far enough to create further reduction. Present capacity of the plant is 240 tons of prepared slag per hr. The plant is operated five days per week.

Slag is eventually loaded into Western Pacific Railroad gondola cars for transportation to points of use. Western Pacific and Asarco have both benefited from development of this new byproduct.

Malay Tin Prospecting May Be Resumed Soon

Malaya must concentrate on prospecting if its tin industry is to continue to produce at the current rate in the next 10 years, according to D. T. Waring, president, Federated Malay State Chamber of Mines.

He said there has been no prospecting since the present emergency began. Mr. Waring expressed concern at the Federation Government's failure to make a comprehensive survey of the tin position as asked for by the Legislative Council in November, 1950.

Mr. Waring reported that tin production during 1952 was 56,830 tons, 329 tons below the previous year. Total dredge output during 1952 showed a decline, but gravel pump mine production grew noticeably.

Meanwhile, General Sir Gerald Templer, high commissioner for the Federation of Malaya, announced arrangements for limited resumption of tin prospecting. He informed company heads that the emergency was so much less severe that he was considering requests for opening areas for tin prospecting.

"If the companies are prepared to take the chance, I am prepared to make some small redeployment of security forces to make prospecting possible."

House Group Favors Tungsten Buying to '58

The House Committee on Interior and Insular Affairs reported favorably on a bill to extend for two years the Government purchase program for domestically mined tungsten.

Representative Wayne Aspinall is the author of the bill. The bill would extend authority to purchase tungsten until 1958. The current law expires in 1956. The General Services program runs until 1956 or until 3 million tons of tungsten at $63 per short ton have been purchased. About 230,000 tons have been bought to date.

Niobium, A Curiosity Once, Now Invaluable

Niobium, of little more than academic interest in recent years, has suddenly become one of the primary objectives of U. S. geologists. They are searching the country for the hard, lustrous, steel gray metal vital in high temperature resistant alloys for jet engines.

U. S. Geological Survey announced finds of rich deposits in a small area known as Magent Cove in central Arkansas. Geochemical survey of the area is under way. Rocks of the region are similar to those from which niobium has been extracted in Germany, Norway, and South Africa.

Niobium was discovered in 1801 by the British chemist Charles Hatchett from rock collected in Connecticut. It was first named columbium. While it is a rare mineral, it occurs in many parts of the world.

Titanium is usually mixed with niobium. Bauxite deposits are also known to have contained niobium. Tantalum has also occurred with niobium. This combination with other elements, of course, has led to difficulty of extraction. Colorado has been found to have some small workable niobium deposits.

Report Algeria Ore Find

Reports circulated in Paris tell of a rich iron ore deposit discovered in Algeria, North Africa, by Mining Exploration Office geologists. Reserves are estimated at 600 million tons. Open pit methods reportedly will be used.

This new Dart truck, designed for underground work, stands only 60 in. high and is powered by a 275 hp diesel engine. Dual controls enable the driver to nose the truck into a work area for mechanical loading, and then swivel his seat forward to drive off. Engine exhaust is rendered harmless by a *scrubber*. The truck weighs 33,700 lb and will carry almost 20 tons.

Iron King mine, owned by Shattuck Denn Mining Corp., and one of the West's most important lead-zinc producers, has been undergoing expansion. The Lorain self-propelled crane handles steel sheets for new thickener tanks at the mill.

Newfoundland to Have Biggest Exploration

The biggest single exploration program in the history of Newfoundland has been launched by the Newfoundland and Labrador Corp., it was announced by H. C. Young, director of mining.

R. J. MacNeill, chief geologist, is in charge of the $300,000 program. Fourteen parties are in the field, nine of them in Newfoundland and the rest in Labrador.

Regions under exploration are the southern tip of Burin Peninsula, Fortune Bay, and Gander River, in Newfoundland. There are also two roving parties. Groups in Labrador will cover the southwestern section, immediately south of the Labrador Mining & Exploration Co.'s holdings, and the Mealy Mountain area on the Lake Melville south shore.

Other firms and the provincial Department of Mines are expected to send men into the field this summer.

Kaiser, Alcoa Contract For Canada Aluminum

Kaiser Aluminum & Chemical Sales Inc. and Aluminum Co. of America contracted with Aluminum Co. of Canada for delivery of a total of 786,000 tons of primary aluminum between 1953 and 1958.

Alcoa will take 600,000 tons, while Kaiser contracted for 186,000 tons. Approximately 75 pct of the total is to be delivered from 1955 to 1958. Announcement of the pact came through Nathanael V. Davis, president of Aluminum Ltd., parent company of Aluminum Co. of Canada.

Fulfillment of the contract may involve "acceleration of construction of further smelting facilities" at Aluminum Co. of Canada's Kitimat development in British Columbia. Authorization has been given for preparation of the ground at Kitimat for two additional smelter pot rooms, with a capacity of about 90,000 tons per year, but decision on actual construction of the smelting facilities and hydro-electric generators is being deferred.

Penn. Railroad Dock At Buffalo, N. Y. Closed

The Pennsylvania Railroad ore dock on the Union Canal, Buffalo, has been closed after two decades as an emergency storehouse for Bethlehem Steel Co. Contract between the railroad and Erie Coal Dock Co., Pennsylvania operating company, has been terminated.

The dock was no longer needed with enlargement of Bethlehem facilities in the Lackawanna slip.

Urge Continuation of RFC Smelter Operation

Continued Government operation of the Reconstruction Finance Corp. tin smelter at Texas City, Texas, was urged by Representative Clark Thompson (D-Texas).

Representative Thompson favors the idea that plants currently operated by the government be turned over to private enterprise. However, in singling out the tin smelter, he noted that the Defense Department terms the operation essential and therefore the time for turning over the smelter to private interests is not yet here.

Senator Harry Byrd (D-Va.) favors the outright abolition of the RFC as provided for in his bill.

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MANUFACTURERS OF QUALITY WIRES AND CABLES FOR EVERY INDUSTRIAL AND COMMERCIAL APPLICATION

THERE is at least one concrete indication that Soviet Russia's two most important iron ore producing centers are seriously depleted. While keeping in mind Russia's penchant for reverse propaganda, one piece of evidence indicates that Magnitogorsk in the southern Urals and Krivoi Rog in the Ukraine have been mined to a point where lower grade ores are being recovered.

An article in *Pravda*, Soviet newspaper, by I. P. Bardin, a Russian metallurgist, says that ore presently mined must undergo an expensive enriching cycle before it can be used in blast furnaces. Compounding Russian metallurgical troubles is the low quality of coking coals. High ash and sulphur content typifies the coal Badin talks about. Coal from the Ukraine, Eastern Siberia, Kazakhstan, and the Far East requires extensive treatment before it can be used, according to the article in *Pravda*.

Russia is in the midst of its fifth Five Year Plan, started in 1951. The program provides for a 32 pct increase in pig iron production. If this goal is to be met, iron ore mining operations must be tripled, Badin says. This could be the usual Soviet straw man, set up to be knocked down by a future gain—paper or actual.

Russia has rich deposits, yet untouched, but far from manufacturing centers. If they are to be utilized, a monumental capital investment may be needed.

DURING May the Reports to the U. S. Atomic Energy Commission on Nuclear Power Reactor Technology were released by the AEC. The reports contained the results of four groups of companies making initial investigations into the feasibility of electric power generated by nuclear reactors. These four groups were: Commonwealth Edison Co., and Public Service Co. of Northern Illinois; Dow Chemical Co. and Detroit Edison Co.; Monsanto Chemical Co. and Union Electric Co.; Pacific Gas & Electric Co. and Bechtel Corp.

The AEC, from the time it took over the atomic energy program in 1947, has searched ways for industry to participate in development of nuclear fission. In 1950, Charles A. Thomas of Monsanto Chemical Co. proposed "that industry might with its own capital design, construct and operate nuclear reactors for production of plutonium and power." The AEC was interested.

Dow Chemical Co. made a similar proposal soon afterward. While considering the proposals, the Commission asked other concerns to offer suggestions. By the summer of 1951 the AEC had arranged for the four groups of two firms each to make at their own expense initial surveys of reactor technology.

The groups were to determine the engineering practicality of their designing, constructing, and operating dual-purpose reactors to produce fissionable material and power; examine the economic and technical aspects of building such reactors within the next few years; determine the research and development required; and submit recommendations as to industry's part in designing, building and operating such reactors.

The companies had available to them pertinent material on reactor technology. Study group personnel were allowed to consult with Commission and contractor scientists and engineers on classified and unclassified information. Twelve months were allotted to the studies. Of course, the reports contained restricted data. Restricted data condensed versions of the reports were distributed in October 1952. The group entered into the work with the knowledge that declassified versions of the work would probably be made public.

Because one of the objects of the study was to determine designs for reactors which might be constructed within the near future the companies for the most part were conservative in their concepts, stuck to "components of proved design, and conditions reasonably certain of early fulfillment."

The four groups were in agreement in the belief that dual purpose reactors are technically possible and could be operated in such a way that plutonium credit would reduce power cost. They also agreed that no reactor could be built in the near future which would be economic for power generation alone.

Commonwealth Edison Co. and Public Service Co. concluded that the heavy water reactor concept holds the best economic promise. However, important uncertainties in the shape of availability and cost of heavy water and practicality and cost of a chemical plant to process fuel elements enter into their thinking. A significant advantage may be had in the use of light water for moderator and coolant if sufficiently enriched uranium is available. Dependability of supply and cost of enriched fuel are uncertainties which may have to be resolved.

It is suggested that a gas-cooled reactor could provide a substantial start in atomic power with minimum interference with production reactors now under construction or in the planning stage. Fuel elements should be cheaper than some used in production reactors and they can be processed in existing chemical plants. Both reactor designs described in the report are considered feasible.

The Dow Chemical-Detroit Edison study reported that three changes were considered in conventional heavy water design. The first modification is a means for increasing the fuel and coolant temperatures of the reactor for practical power production by the use of liquid-metal coolants. Another change was development of a new type of metallurgical separations process. Another alteration was modification of the fuel system of the reactor to allow for closer integration with the separations process. The report states: These changes should be capable of incorporation in the heavy-water or graphite reactors, but the time and expense involved are considered to be as great as that required to develop the fast breeder reactor with its low cost and virtually unlimited availability of fuel.

Monsanto Chemical Co. and Union Electric Co. began their investigations with the Hanford type reactor with the understanding that certain modifications would have to be made in order to convert

the heat produced into commercially saleable power. Certain advantages were revealed for sodium as a high temperature coolant. Sodium-potassium eutectic, however, had the advantage over sodium of a lower melting point and thus a lower possible inlet temperature with a higher possible heat pickup.

Cost of reactor-produced electric power was found close to that from a modern coal-burning plant. It costs more to absorb heat by making steam acceptable for a turbogenerator than to dump the heat into a river. Also, the short five year amortization period assigned to reactor and power plants raises depreciation charges to a level, when combined with other cost increases, that offsets savings in fuel.

Pacific Gas & Electric Co. and Bechtel reported estimated costs for a 106.2-megawatt water-cooled thermal reactor at $41 million and for a 154.6-megawatt sodium-cooled fast reactor at $51 million. Total estimated yearly operating cost for the water-cooled reactor is figured at $1,580,000 and for the sodium-cooled reactor at $1,520,000.

North American Aviation Inc., in the meanwhile, disclosed a design for a 8000 kw pilot plant to furnish information and engineering data. North American states that the pilot plant would take about two years to build and would furnish data to construct a 150,000 to 250,000 kw plant for about $300 per kw compared with $200 per kw for a coal plant.

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THE Brooklyn Bridge! Who hasn't heard of it? No matter where one travels on this earth, there is sure to be someone who knows of it. Poets, novelists, and painters without number have rhapsodized over its dignity, majesty, beauty, and grace. It was New York City's first link with the fabulous land of the Brooklyns, built during the era of gas lights and buggies, and when Queen Victoria reigned over the British Empire. Last May 24 the Brooklyn Bridge was 70 years old. In recent years once in a great while a horse and wagon could be seen plodding across the span amidst the mass of 20th century speed racing across in either direction. Today, the old girl is getting a facial, a new set of stays, and a general rejuvenation.

When John A. Roebling built the bridge, he had to create the design, form a manufacturing firm to make the cable, and then act as engineer, foreman, and do whatever else was needed to get the job done. John A. Roebling's Sons Inc., now a subsidiary of Colorado Fuel & Iron Corp., is still building bridges in every corner of the globe.

The Brooklyn Bridge was the first suspension bridge to use steel wire rope, invented by Mr. Roebling. Engineers recently declared the galvanized steel cables supporting the bridge to be in perfect condition. It was decided to modernize the bridge and when it is reopened at the end of the year it will have two three-lane highways to handle the tremendous traffic going over it daily. There will also be a bronze tablet commemorating the woman who made the bridge possible—Mrs. Emily Warren Roebling. She fought by her husband's side, beating back political opposition until the bridge was finished.

When the Bridge was first opened to traffic in 1883 rumor spread like wildfire that it was unsafe—that it would fall into the East River. It seems the Lady will be around for a long time yet.

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SOME Pennsylvania soft coal mine operators see no hope for the small mine in the next five years. When the Ford Collieries Co. closed in the Allegheny Valley, it brought to 15 the number of mines closed within the last 18 months. Mounting costs, decreasing markets, competition, and other economic factors are blamed for the closing. One operator put it this way:

"Conditions will get worse in the next five years. The day of the small nonmechanized mine is drawing to a close. They cannot compete with the large commercial operations. These, along with mines operated by steel companies which consume their own output, will hold the field. What the future holds cannot be predicted but there is no immediate relief in sight to bring business back to coal."

Would mechanization help? Perhaps, but one of the basic troubles with small mines today is that operators can't afford to mechanize. Can anything be done? John P. Busarello, president of the United Mine Workers District 5, states that two mines nearing the shutdown stage were saved when a plan was contrived by union and management to boost productivity per man, making operating costs lower.

But a combination of causes seems aligned against the western Pennsylvania coal operators. They claim competition of heavy imports of cheap foreign oil waste, which is supplanting steam coal as an East Coast industrial fuel, is one item. Western Pennsylvania operators point out that their operating costs are higher than those of West Virginia. Nonunion production in several Ohio and Kentucky pit and strip mines is also cutting into western Pennsylvania markets. These factors, combined with railroads switching to diesel power, UMW Welfare Fund, and depletion of strip operations, are blamed by many operators for the current situation.

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"DON'T TOUCH A BLASTING CAP" is the warning on posters to be displayed in 41,000 U. S. post offices as the Post Office Department and the U. S. Bureau of Mines join forces in the summer safety campaign. Posters show four types of blasting caps and warn youngsters not to touch them. "Tell a policeman about it." Summer is probably the most acute time for this kind of danger. Because summer is a peak construction season, more blasting caps are used and thus more likely to be found when carelessly discarded.

Drift of Things

A NOTED Prospector Passes—Stories of the grizzled old prospector that discovered what turned out to be a famous and highly profitable mine, only to die in poverty through the shenanigans of "the Interests," are legion. It is pleasant, therefore, to be able to report that the discoverer of one of Canada's most famous mines was more fortunate.

E. H. Horne staked the original claims of what is now the property of Noranda Mines, in northwestern Quebec, in September 1922. He was grubstaked by what was known as the Lake Tremoy Syndicate, in which he and his fellow prospector had a 20 pct interest. Ed Horne was a Nova Scotia farm boy who became a wiry, rawboned prospector, roaming the wilds of North America from Labrador to Nevada and California and north to the Yukon looking for gold. At the age of 60 he hit the jackpot—not gold but copper. He sold his interest for a reputed $280,000, married, and returned to his native Enfield, Nova Scotia. There he raised one of the finest Hereford dairy herds in Canada. On March 15 last he died, at the age of 87.—*E.H.R.*

ON page 446 of the May, 1953 issue of MINING ENGINEERING, we published a letter from a colleague in Korea. Responding to this letter and personal correspondence, we received the following from him dated April 25. At that time it caused considerable stir among those that read it. Here it is:

Dear Mr. C. M. Cooley:

It was very pleasant to read your kind answer dated Feb. 10, and I am obliged for having caused you the trouble of publishing my poor letter in your magazine, and to have promised that you would make efforts to get books for me. May I ask you to send me the volume in which the letter is published?

You see, the cease-fire council has been held between U.N. and the Reds nearly two years in Korea, and recently it showed activity through exchange of sick POWs. You may, however, be astonished to hear that we Koreans have a bitter opposition to it, because the truce without national unity is so dishonorable for us. We hope the day will soon come when our national unity is completed under the democratic policy.

South Korea has more than fifty colleges, but only four or so have a mining and metallurgical course. I am sorry none of them has a laboratory to deserve the name. I think it is my duty to import your recently developed technics on mining and metallurgy, teach them to our students and contribute to the mining industry in Korea.

I am very glad to have found a good leader, and hope you will guide me forever. Please let me hear the news on modern development in milling.

Hoping that you are still enjoying good health.

Yours respectfully,
Kang Moon Lee

FOREVER Is a Long, Long Time—Looking back through some of the old records of the AIME recently, we came across a resolution of the Board of Directors at their Meeting on April 26, 1918. Feeling against enemy aliens was much greater at that time than at any time during World War II, we believe. The resolution passed by the Board reads as follows: "That all Honorary Members, Members, Associates, and Junior Associates who are Enemy Aliens be dropped from membership in the American Institute of Mining Engineers and their names be stricken from the rolls, forever."

Among the casualties of this Resolution were three Honorary Members: Richard Beck and H. C. Emil Schroeter, of Germany, and Hans Hoefer, of Austria. Their names never again appeared in Institute Directories, either as current or deceased Honorary Members. When World War II broke, the Institute had no Honorary Members in enemy countries, nor have any been elected since.—*E.H.R.*

ANDY Anderson, hard-rock miner who is walking his burro, Bosco, from Los Angeles to Fairplay, Colo., recently passed through New Mexico, according to the *New Mexico Miner*. Miner and burro are advertising the fifth annual running of the Rocky Mountain Pack Burro Race, August 2. Participants face a monumental task, and the words "race" and "running" are used advisedly, the western pack burro being basically conservative.

Contestants must push over 13,000-ft Mosquito Pass on the route leading from Leadville east up past the Ibex mine, following the old Mosquito Pass road over the range, dropping down past the London mine on the eastern slope and into Fairplay. Mosquito Pass is passable now only to burro and Jeep, but in the 1880's it was the regular route for the Alma to Leadville stagecoach.

Bosco, like the other entrants, will carry a standard prospector's pack which must include pick and pan. Time for the 16-mile up-down route is amazingly short, about 4 hr 15 min in previous races. Having been in the district when the first three races were held, we were surprised and pleased to see how far this event had gone, because at the inception the burro race was purely a local affair.

FROM the *Journal of Commerce*, a story with or without a moral: "The grizzly bear was down in the canyon behind a tree," the old prospector was saying. "The only way I could hit him was to ricochet a bullet off the high canyon wall on my right. Well, I gauged my windage, calculated the lead of the barrel and the rate of the twist, the hardness of the bullet and the angle of the yaw it would have after being smacked out of shape against the canyon wall. Then I fired." His listener asked: "Did you hit him?"

"No," said the old timer, "I missed the wall."

Charles M. Cooley

Recent changes at Cornwall are highlighted by development of a panel caving method, with slushing drifts, for underground practice as open pit operation nears close. Metallurgy keeps the pace with high recovery from increasing amounts of harder to liberate, higher sulphide content, underground ore.

After Two Centuries of Mining Cornwall Keeps Its Methods Up-to-Date

by S. J. Shale

SINCE 1742 the first iron orebody at Cornwall, Pa., has been mined continuously. The mine supplied ore to the charcoal furnaces which made cannon and ammunition for General George Washington and his Continental Army during the American Revolution. The original Cornwall charcoal furnace, built in 1742, is still in excellent repair and is now a State owned museum.

Located in Lebanon County, 6 miles south of Lebanon, Pa., the mines are operated by the Bethlehem Cornwall Corp., a Bethlehem Steel Co. subsidiary. Two major lenticular orebodies, about 3000 ft apart, are being mined. The first known orebody has been mined by open pit methods from the earliest days, and lower portion of the orebody, not recoverable by open pit methods, is served by the No. 3 inclined shaft. The second orebody, or No. 4 mine, which lies entirely underground, was discovered in the early 1920's, although ore requirements did not necessitate bringing it into full scale production until 1938.

Magnetite deposition at Cornwall is presumed to have resulted from contact replacement of Paleozoic limestone by high iron bearing solutions associated in some manner with late Triassic intrusion of diabase trap rock. The formations have a general east-west trend and dip to the south with a footwall of trap rock and capping of slate, conglomerate, quartzite, or limestone. The ore, containing about 42 pct magnetic iron, also carries important recoverable by-products of chalcopyrite which bears a small amount of gold and silver, and pyrite containing 1.25 pct cobalt. It is believed that until recently Cornwall was the only source of cobalt in the U. S.

Open Pit Operations

In the early years of the pit ore was transported by wheelbarrows, and later, by horse drawn carts, to the neighboring charcoal furnaces. As the pit became deeper and the demand for iron ore greater, steam shovels and steam locomotives were used. Railroad pit haulage was in turn replaced by truck haulage in 1944, because of depth of operation and attendant steep grades.

Mining is accomplished by breaking an 18-ft bench along the pit bottom. Churn drills were used for blast hole drilling until 1949, but since then only wagon drills have been used. Tungsten carbide bits are used exclusively, averaging about 2900 ft per bit with drilling rates of 2 to 3 fpm in ore. Blasting is done electrically, with about 65 lb of dynamite per hole. About 3 tons of ore are broken per lb of explosives used.

Broken ore is loaded by two electric shovels into 30-ton capacity, side-dump trucks which transport ore to the primary crusher, located within the pit, where the tandem 10-wheel trucks are dumped by a stationary crane. All six of the 200-hp trucks have hauled over a million tons each.

A primary 56x72-in. jaw crusher reduces ore to −12 in. Crushed ore is hoisted in two 12-ton capacity skips, operating in balance, to the secondary crushing plant located on the surface, where a series of roll crushers reduce the ore to −1¼ in.

Average daily tonnage in 1951 was 2960 tons per day on a single shift basis.

No. 3 Mine

No. 3 shaft was sunk beneath the west end of the open pit to mine ore not economically recoverable by open pit methods. The inclined shaft, completed in 1922, was sunk 1600 ft in the footwall, at an average dip of 26°, and about 100 ft below the ore in the trap rock.

Underground mining started at the lower extremity of the orebody and progressed up-dip, utilizing shrinkage stoping. Stopes were drawn through a system of breaking subs, grizzlies, and transfer raises. Operations were suspended in 1941 to prevent interference with open pit operations. Recently, with open pit operations nearing completion, the mine was reopened and is being developed for panel caving, a method worked out at No. 4 mine, and the

S. J. SHALE is Manager, Bethlehem Cornwall Corp.

Looking east into Cornwall open pit, continuously mined for 211 years.

one that has proved best for Cornwall ores. Upon completion of the open pit in 1953 No. 3 mine will go into full production at about 2500 tons per day.

No. 4 Mine

The orebody has the shape of a giant lima bean. In plan view it is 2000 ft wide; in section it is 235 ft thick at the middle and pinches out at the extremities. Lying within 150 ft of the surface, it dips south at 30° to a depth of 1200 ft.

No. 4 mine is served by the No. 4 and No. 5 inclined shafts. About 220 ft apart, the shafts were sunk on a catenary curve in the trap rock footwall about 100 ft below the ore-trap contact. The slope at the collar is 36°, flattening to 26° at the bottom. No. 4 shaft, used for hoisting ore and rock, is equipped with an 11-ft diam, cylindrical, 2-drum hoist using 1½-in. cables and operating at 1200 fpm. No. 5 shaft, used for handling men and supplies, is equipped with a 6-ft diam, cylindrical, 2-drum hoist using 1⅛-in. cables and hoisting at 600 fpm for men. Each shaft is 7x20 ft with three compartments, two skipways and a manway.

Shaft sinking started in November 1924, and by 1926 No. 4 was sunk 1470 ft along the incline, or 800 ft vertically. Drifts at the 300, 500, and 700-ft levels were driven from No. 4 to the No. 5 shaft location, and the No. 5 shaft was excavated by driving pilot raises simultaneously between the three levels and the surface. These raises were subsequently enlarged to 7x20 ft and supported with steel and concrete. In 1947 shaft sinking was resumed, and both shafts completed in 1948 to a vertical depth of 1225 ft.

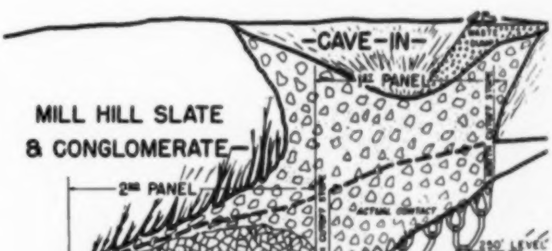
Since accurate ore-footwall information is vital in the location of the development openings, exploration drifts on alternate levels are driven in ore 50 ft above the footwall contact, along the strike, to the extremities of the orebody. Footwall and hanging wall information is obtained by diamond drilling the exploration drifts with a series of radial holes, in a plane at right angles to the strike, on 100 ft coordinates.

Until 1931, when the mine was closed due to the depression, development was patterned after the method used at No. 3 mine. At No. 3 mine, shrinkage stopes were mined over a row of bells (or mills), between haulage levels and from footwall to hanging wall. Two shrinkage stopes 13 ft wide with a 7-ft pillar between were mined simultaneously with one stope well ahead of the other. As the lower stope advanced, the pillar was drilled and blasted. Ore was drawn out through a system of breaking subs, grizzlies and transfer raises.

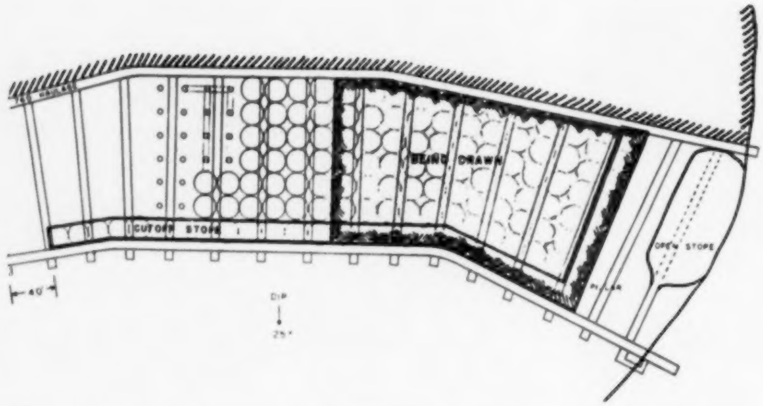
When No. 4 mine was reopened in 1937, it was decided to try a block caving method of mining instead of the shrinkage stope and pillar method. However, the system of breaking subs, grizzlies and transfer raises was retained. This is shown for the 250 and 300-ft levels in the diagram below.

Belling commenced at the east end of the orebody and retreated toward the shaft to undercut an area 170x415 ft. This panel of ore was previously cut off on three sides by shrinkage stopes. The area was drawn nearly empty without any noticeable caving of the hanging wall. Many large ore chunks were encountered over the bells because of the lack of hanging wall pressure and breaking these chunks required a great deal of secondary blasting. The largest chunk encountered was approximately 70-ft diam and weighed about 20,000 tons. A raise with drifts was driven in the chunk and 800 blast holes were drilled from the drift to break the chunk.

At this time it was decided to put down 6-in. well drill holes from the surface into the hanging wall to induce caving of the hanging wall rock. It was feared that if a larger area was undercut and caving followed, a damaging air blast might be produced. Two holes were drilled to within 15 ft of the hanging wall. These holes were blasted without any



Block caving, with breaking subs, grizzlies and transfer raises is shown on 250 and 300-ft levels. The 400 level uses new system with slusher drifts.



PLAN of workings used in caving with slushing drifts. Development is carried forward from east end of orebody, progressing toward the shaft. Haulage ways are concreted.

apparent effect on a Saturday. The following Tuesday morning, the back started working and intermittent falls of rock were heard during the day. Subsidence to surface was noted at 11 pm. There has been no difficulty encountered with the caving action since that time.

Greatest single weakness of the breaking sub method is the intricate raise system which honeycombs the trap rock just below the breaking subs. Since the blocky trap rock will not stand wear or pressure, much trouble was encountered in maintaining the raise system. Therefore, later in 1937, it was decided to develop future levels for panel caving, using slushing drifts instead of breaking subs. The following anticipated advantages have been realized:

1—Easier draw point maintenance.
2—Faster and cheaper development.
3—More flexible operation.
4—Safer working conditions.
5—Slushing drifts driven in the trap rock footwall parallel to the contact maintain draw points at a desirable distance below ore contact.

The only disadvantage is that water coming in one or two bells can wet all the ore drawn through a particular slushing drift. However, as ore draw retreats, a major portion of the water is left behind in the emptied area.

A panel of ore (one complete level) approximately 150x2000 ft is developed in the following manner: A cross-cut is driven from the shaft to the predetermined location of the haulage drift. The cross-cut is 100 ft below the contact at the shaft and it is driven toward the orebody until it is only 30 ft below the contact.

The haulage drift is driven parallel to the strike in the trap rock footwall to the extremities of the orebody, always keeping 25 to 35 ft below the contact. The grade in favor of the load is ½ pct. A 3-machine hydro-jumbo, mounting 3½ in. drifters with 4-ft feed, is used for driving haulage drifts. Tungsten carbide detachable bits are used exclusively, drilling 9 to 13 in. per min in trap.

Muck is handled by a pneumatic shovel loading a conveyor belt which discharges into mine cars. The 8-ft deep drift round is 9x9 ft with an arched top and a 3-man crew can muck out, drill, and blast a round each shift.

The drift is lined with concrete, which is always placed before any further development is started, because the trap rock is blocky and must be supported as soon as possible. After concreting, the arched top haulage drift is 8 ft wide, by 7 ft 6 in. high and has 30-in. gage track using 60-lb rail.

Development work starts at the east end of the orebody and progresses toward the shaft. Slushing drifts on 40-ft centers are driven above and normal to the haulage drift, parallel to the contact, maintaining as closely as possible an average distance of 35 ft between the floor of the drift and the contact. Average grade is 25 pct with maximum safe working grade of 35 pct. Slushing drifts are about 150 ft long, and break through into the haulage level above, providing good ventilation for secondary blasting.

The arched top slushing drift is driven 8x8 ft, with two drifters mounted on one column, using tungsten carbide detachable bits. Two shifts are required for the complete cycle of each round.

The slushing drift is lined with concrete after it is completely driven. Finger openings, 3 ft 6 in. x 4 ft 6 in. on 20-ft centers, are formed in the concrete on both sides of the drift. The finished drift is 6x7 ft with an arched top and has six or eight finger openings on each side.

Finger raises are driven vertically through the footwall, and one round into the ore. Average finger height is 35 ft but varies according to irregularities in the dip of the footwall contact. Raises are 10 ft from the center of the slushing drift.

A manway drift connecting the tops of the finger raises is driven in ore on each side of the slushing drift, starting at the front finger and progressing up-dip parallel to the slushing drift, and connected at the upper end. The drift round is drilled with a hand-held jackhammer and single-use bits.

Reference to the diagram shows the procedure for cutting off the panel of ore being developed. The front fingers of several slushing drifts are enlarged to bells and an east-west shrinkage stope is mined. The stope is 6 ft wide and may extend to the hanging wall if the ore is thin, but rarely exceeds 60 ft height. The panel of ore has already been cut off on the north by the mined out area and on the east by an open stope or another shrinkage stope.

After the east-west shrink is complete, all the other fingers are enlarged to bells. This work proceeds up-dip by belling pairs of fingers in each drift. The bell is worked on a shrink method. Rows of holes are drilled around the raise, fanning upward and outward and are blasted. Broken rock is left in the raise and entrance is gained by using the manways in ore and entering through the top. Enough broken rock is drawn after each shot to maintain a floor suitable for drilling up-holes with a stoper. The holes are fanned out in each succeeding round,

Undercut area at top of bells in caving method.

so the finished diameter at the top of the bell is 20 ft.

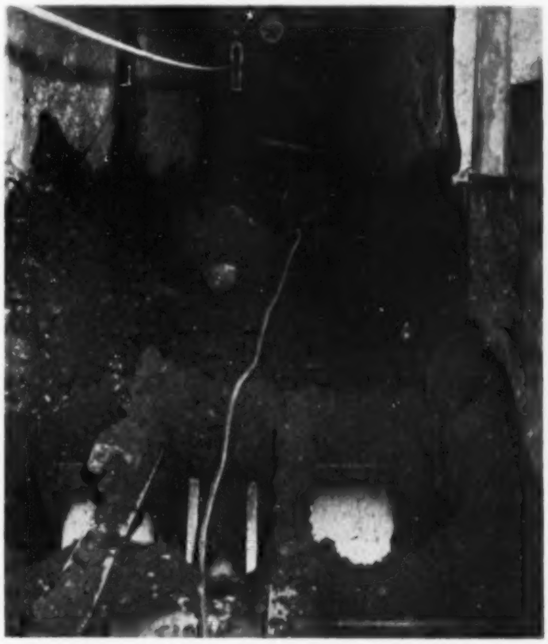
The pillar, which would remain between adjacent bells, is removed by drilling long holes and blasting it with the last round of the bell. About 17 man-shifts are required to complete one bell.

The ore, which has been cut off on three sides and is being undercut with bells, forms a cantilever. As the length of undercut on a new level approaches 240 ft, drawing can be started and the cantilever effect will cause the ore to continue caving. Development continues across the entire orebody, but at least 80 ft is maintained between the area being drawn and the area being belled. The hanging wall caves readily to the surface on about 65° plane.

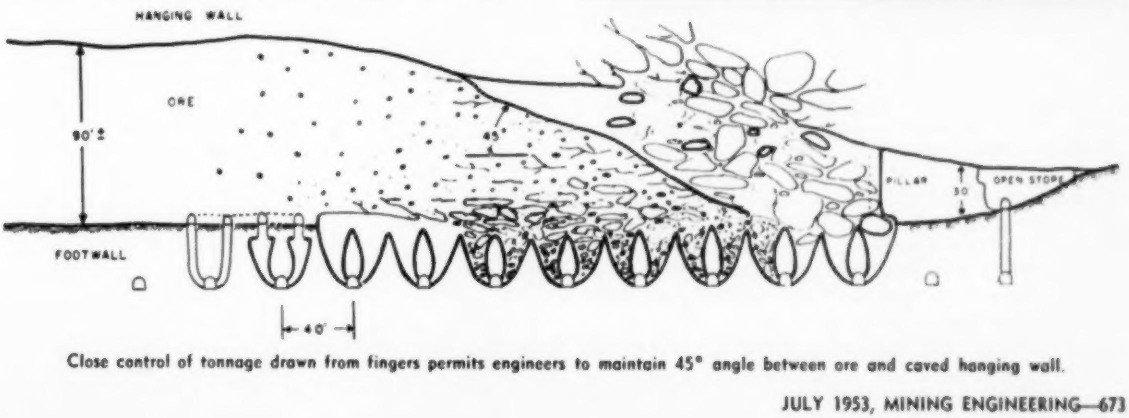
Ore drawing usually starts as soon as 240 ft of a new level is developed. Each slushing drift in production has a 2-man crew who draw the fingers and scrape the ore into the cars. A 50 hp, double drum, electric hoist with a ¾-in. pull rope and a ⅝-in. return rope pulls a 54-in. semi-box type scraper. Frequent blasting is necessary to break chunks that block the finger openings. One third of all the dynamite used underground is consumed by secondary blasting, amounting to 0.14 pounds of explosive per ton of ore mined.

The tonnage of ore available at each finger has been previously calculated from exploration drilling data. A draw control engineer determines which fingers are to be drawn and how much ore is to be drawn from each finger during each shift. Accurate daily records enable the engineer to maintain the desired angle of 45° between the plane of the broken ore and the caved hanging wall. This minimizes funneling with subsequent dilution of the ore by

View of concreted slushing drift shows 54-in. scraper.



RIGHT: Troublesome branch raises are eliminated by 90-ft scraping trench serving three levels. Ore is delivered to measuring hoppers by 125-hp hoist pulling a 72-in. scraper.



Close control of tonnage drawn from fingers permits engineers to maintain 45° angle between ore and caved hanging wall.

hanging wall rock. It also eliminates the possibility that some ore may not be recovered because the capping rock has worked underneath and cut it off.

Two levels of the mine are always in full production with a third level nearing completion. Three or four slushing drifts are usually being drawn on each shift on each level. Production from each slushing drift will average 150 net tons per shift or 600 net tons per level per shift. During 1951, No. 4 mine operated 306 days and averaged 4384 net tons per day after cobbing, equivalent to 12.1 tons per man-shift for the entire force, surface and underground.

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Two methods for loading hoisting shaft skips are in use. Both methods use measuring hoppers from which the measured load of ore is dropped into the skip by an air-operated flap-gate, and the difference lies in the method of filling these measuring hoppers. The older system has air operated arc gates at the bottom of a single raise, fed through concrete and steel lined branch raises from the three haulage levels above. Serious troubles were encountered with this system. Abrading of the rock at the points where the raises branched into each other presented costly and dangerous problems. A hang-up in the lower section held up production from all three levels. Dislodging the hang-up often resulted in a rush of wet ore past the loading pocket down into the shaft. As a result of these problems the loading system was redesigned for a scraping type pocket.

The scraping type pocket has a scraping trench 90 ft long into which separate raises feed from each of the three levels above. Ore is scraped through the trench into the measuring hoppers by a 72-in. scraper, powered by a 125-hp electric hoist. With this system, troublesome branch raises are eliminated and a hang-up in one raise affects only one level. If a rush of wet ore occurs, it will stop in the scraping trench without spillage into the shaft. Since only one third of the total ore handled in the pocket passes through each raise, it is apparent that raise maintenance will be less than encountered in the former system where all three fed into a single raise. The scraping trench system has been found to be superior to the previous system and all new loading pockets will be patterned after it, although some pockets will service only two levels.

Ore and rock are hoisted in two 10-ton skips, hoisting in balance at a speed of 1200 fpm. Approximately 170 skips per eight-hour shift, or 510 skips per day are required to maintain production. The skips dump into a 600-ton bin in the headframe.

A 7-ft diam roll feeder feeds ore to a 36x72-in. jaw crusher which crushes to −3 in. A 48-in. conveyor, traveling at 300 fpm, carries the jaw crusher product to the secondary plant. This conveyor has a magnetic head pulley which separates the ore and the waste rock. The waste rock, consisting of dilution from the hanging wall and footwall development rock, is passed over another short conveyor belt which also has a magnetic head pulley to retrieve any ore not picked out by the first belt. In 1951 waste rock averaged 10.6 pct of the total tonnage hoisted.

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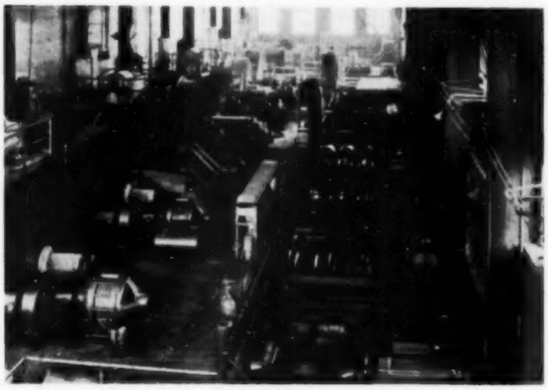

Mill section at concentrator in Lebanon.

Mill Feed

Open Pit Ore	Underground Ore (No. 4 Mine)
42 pct Fe	40.3 pct Fe
0.2 pct Cu	0.3 pct Cu
1.1 pct S	1.3 pct S

Iron Concentrate

Fe (total)	62.4 pct
P	0.007 pct

Flotation Products

Chalcopyrite Concentrate	pct	Pyrite Concentrate	pct
Cu	25.0	S	50.0
S	30.0	Fe	44.0
Fe	30.0	Co	1.27

Tailing

Fe (total)	7.20 pct
Fe (magnetic)	0.30 pct

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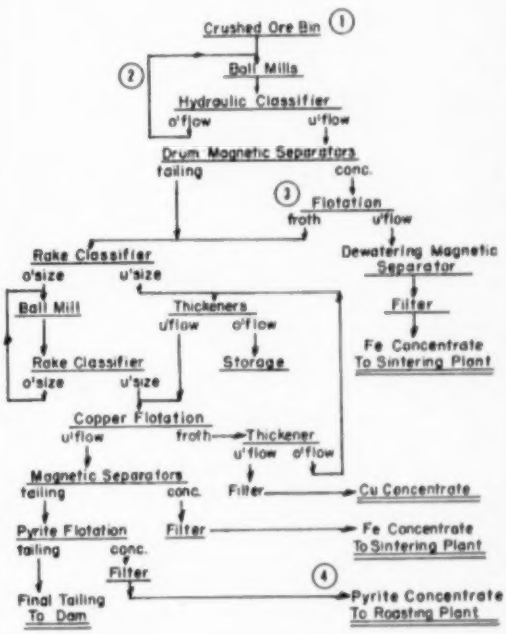

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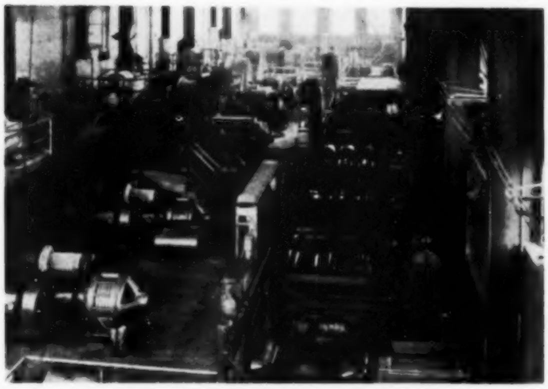

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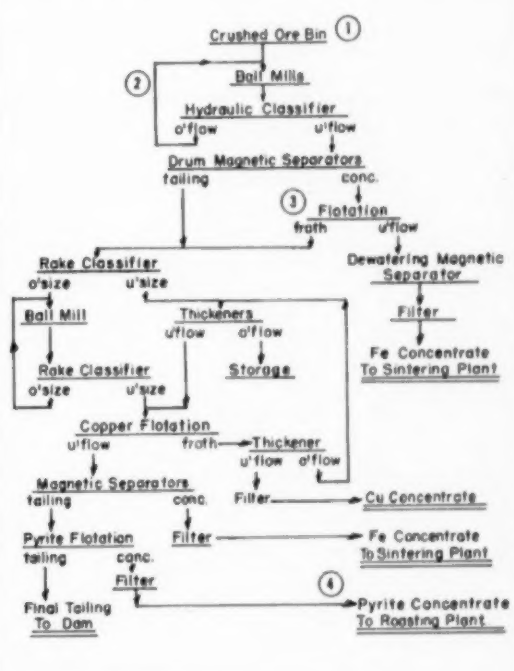

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Cleveland-Cliffs Conveyor System Uses Twisted Belt To:

1—Beat Upper Peninsula Weather

2—Solve Underground Transportation Problems

by D. Kelly Campbell, H. H. Korpinen, and H. C. Swanson

DESPITE defiant Michigan weather, engineers working at the Mather mine "B" shaft decided that belt conveyor stockpiling could be used providing problems of low temperatures and sticky ore were solved. Belt conveyors presented definite economic advantages over the railroad trestle system used in the "A" shaft. However, gains could easily be offset by operating difficulties.

Wet ores and below zero temperatures at the Cleveland-Cliffs Iron Co.'s Upper Peninsula operation could cause ore to adhere to the conveyor belt and to build up on the return idlers. Ore build-up spells excessive spillage, belt misalignment, and costly maintenance and repairs. By simply twisting the return flight conveyor at both ends, the answer was found. The clean surface of the belt is always presented to the idlers, spillage is concentrated at the point of twist, and mechanical maintenance is minimized.

Lester Ericson of National Iron Co. conceived the idea of twisting the belt. National Iron cooperated with B. F. Goodrich, Chain Belt Co., and Cleveland-Cliffs in test work. Detailed results of these tests and mathematical solution of design problems appeared in MINING ENGINEERING, December 1951, written by J. W. Snavely.

D. K. CAMPBELL, H. H. KORPINEN, and H. C. SWANSON are with Cleveland-Cliffs Iron Co., Ishpeming, Mich.

Success of experiments with twisted belts is evident from the number of installations Cleveland-Cliffs is using:

- June 1951—Mather mine "B" shaft; main stem surface stockpiling conveyor; 655-ft head to tail pulley, 36-in. width, 400 fpm.
- January 1952—Mather mine "A" shaft, underground conveyor; 2600-ft, 30 in., 500 fpm.
- March 1952—Mather mine "B" shaft, east branch stockpiling conveyor; 450-ft, 30 in., 500 fpm.
- March 1953 (schedule completion)—Mather mine "B" shaft, west branch stockpiling conveyor; 1185-ft, 30 in., 500 fpm.

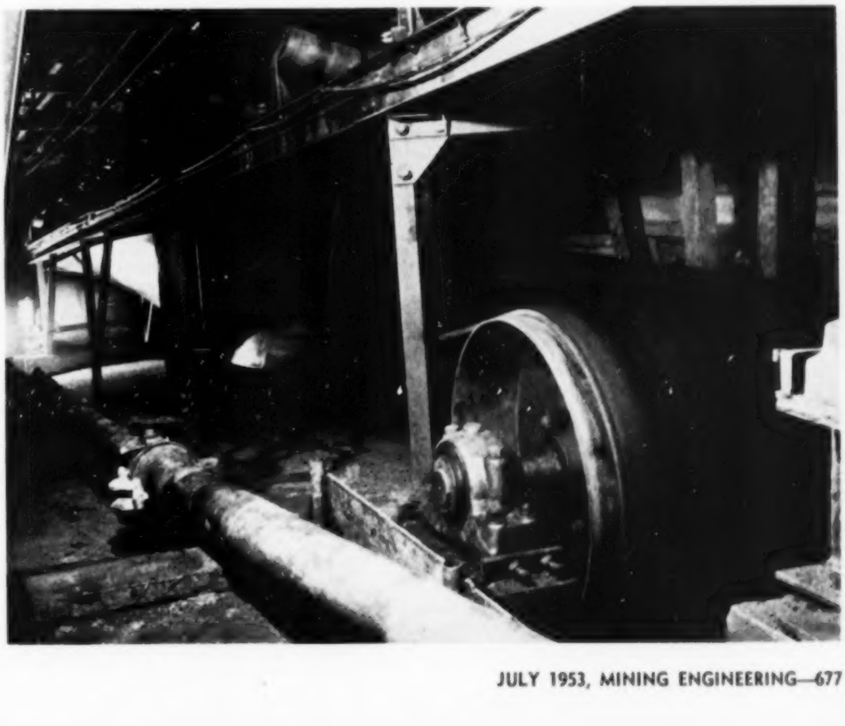
Experience with the first stockpiling belt at the "B" shaft led to the installation of the new underground belt conveyor at the "A" shaft, incorporating the twisted return flight feature. Stepped-up production on new levels requiring overlong tramming distances led to installing the belt on the seventh level. A 7x7 drift was driven for the belt from the shaft parallel to the main haulage level. The conveyor transports ore 2500 ft horizontally and elevates it 85 ft, from a point 35 ft below the haulage level to 50 ft above the shaft loading station.

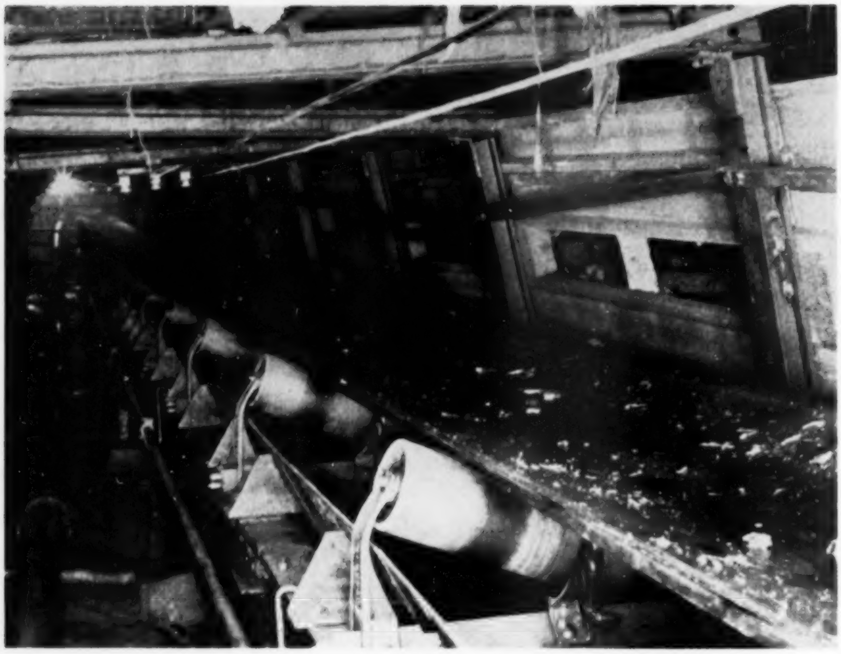
Top side of "turnover" belt does its job underground where ore is very wet, sticky, and tends to build up on ordinary belt installation. Illustrations on opposite page and below show the twist that makes the tunover belt unique.

Ore is gathered in cars from chutes at the transfer raises and trammed by locomotive to the crusher station, where it is automatically unloaded by ramp car dump to a scraper trench. Ore is then scraped over a fixed grizzley with 6-in. openings to a 42x36-in. crusher. Crushed ore joins fine material from the grizzley at the pan feeder. The pan feeder discharges to the belt conveyor at approximately 35 ft below the haulage level. The conveyor transports the ore to a point 50 ft above the shaft loading station. Three raises controlled by a butterfly gate arrangement transfer ore to a scraper trench feeding the 12½-ton skip measuring pocket. These raises provide storage space and permit segregation of ore from waste rock. Production from the seventh level is expected to reach one million ton mark sometime during 1953.

The diagram presents details of the conveyor installation. The belt is twisted 180° at head and tail ends when installed and prior to vulcanizing. First twist is 180° in one direction and the second twist is in the same direction. This 360° rotation will

Twist is put in "turnover" belt in this 36-ft span underground. Return strand of belt, at bottom, moves right to left with carrying surface turned 180° and brought uppermost at extreme left.





Unloaded belt in underground installation shows clean surface despite wet, sticky ores handled. Absence of spill is another feature; surface installation didn't require cleanup after a year's operation.

equalize any stretch in the belt, because edge stretch in first twist will be compensated by second twist.

Pulleys in the conveyor system can be adjusted horizontally and are used in training the belt after installation. While vertical adjustment would perform the same task, horizontal adjustment is found simpler.

It was necessary to install two vertical guide rollers at the midpoint of belt twist at the Mather "B" shaft stockpiling system. The belt at this point is vertical and because the full width is presented to the roller surfaces there is little wear. Vertical rollers were necessary because right angle belt loading and variation of ore from wet to dry caused the belt to float laterally along the width of the tail pulley. Right angle loading of wet ore placed the bulk of the material on the near side of the belt, causing the belt to track toward the far side. Dry material, with greater trajectory, was deposited on the far side and the belt tracked toward the opposite side. Installation of vertical guide rollers entirely corrected wandering of the belt across the tail pulley fall.

The loading condition which necessitated the development of vertical idlers does not exist at the discharge belt at the Mather "A" shaft. However, the vertical rollers are a positive aid in training the belt and providing a safeguard, aligning the belt under any unusual conditions. John Nigra and Geno Colombo designed the vertical guide rollers at the Mather "B". All Cleveland-Cliff's twisted belt systems incorporate the device.

Deck plates between the carrying and return belts have been eliminated in the conveyor structure and diverting skirts added in their place. This directs any spill from the loaded belt to the return flight. The return flight is equipped with troughing idlers instead of flat idlers, assuring carrying back spill to the rear turnover points immediately preceding contract with the tail pulley.

Initial installation of the twisted return flight belt conveyor was made at the "B" shaft on the 655-ft stockpile conveyor. The belt runs north from the crusher building to the stockpile area. This is the main stem conveyor of the system, from which one lateral conveyor extends 450 ft east. A 1185-ft belt will be completed on the west side this summer. Although the main belt is 36-in. wide and is driven at 400 fpm by a 60-hp electric motor, the lateral conveyors are handling the same amount of ore on 30-in. wide belts by speeding up to 500 fpm.

The belts elevate the ore approximately 35 ft but stockpiling to a height of 65 ft can be achieved because the terrain slopes away from the collar of the shaft. Winter stockpiling capacity at a rate of 5000 tons per day will be provided when the system is complete.

The conveyor is completely enclosed in a structure specially heated and insulated to prevent ore from freezing and sticking to the belt. Enclosure walls are corrugated galvanized sheet on the inside, with 2 in. of 7-lb density fiber glass between the inside galvanized sheet. The moisture proof outside wall is of protective-coated corrugated sheeting. The heating system uses a 15,000 cfm fan to force air through low pressure steam coils. Approximately one third of the heated air is forced directly through the enclosure under the belt, with two thirds being exhausted into the gallery. It was found that the forced air system is only needed when the atmospheric temperature drops below 20°F. Ports, protected by outside sliding doors and hoods in the sides of the conveyor enclosure, discharge ore to the stockpile. The sliding doors are insulated. Dogs and a rubber seal prevent loss of heat.

The twisted return flight feature eliminated the necessity for cleanup along the length of the main stockpile belt. The floor is clean after a year's operation, although it hasn't been swept. Cleaning is needed only at points where the belt is twisted and at transfer and discharge trippers. Idlers haven't been changed to date.

Financing Domestic Mining Ventures

Part II Sources of Capital Funds

by C. C. Bailey, L. C. Raymond, and W. F. Boericke

THE vendor seeking outside capital for his project has the following potential sources which may be approached;

1—private sources,
2—institutional funds,
3—government agencies,
4—established mining concerns, and
5—public financing.

Private Sources

Capital available from individuals is usually limited by the range of personal contacts available to the vendor. Moreover, it is usually feasible to seek capital from individuals only in early stages of a mining project when the amount of funds needed to accomplish the next step is small, within the range of $50,000 to $100,000.

Institutional Funds

Institutional funds (such as might be supplied by insurance companies, commercial banks, endowments, etc.) are in general only available for mining projects when there is large and long term operation controlled by interests of demonstrated experience and competence. Furthermore, the fiduciary character of such funds practically requires a preferred status in the capital set-up of the project, which means that they must be represented by bonds and notes, rather than preferred and common stocks. Under these circumstances, in order to attract institutional funds, it is necessary that the project have a substantial equity investment to protect the senior liens, or substantial assets of the controlling interests for adequate supporting collateral.

The costs of securing institutional funds are borne by the vendor out of equity funds already in the treasury or secured as additional equity financing. The senior lien will have a fixed maturity date, usually from 10 to 25 years, since institutional funds would not be interested in smaller short-term projects. The interest rate would fall between the limits of 3 to 6 pct per annum, and there would be sinking fund provisions designed to retire all, or the major portion, of the loan by maturity.

A current example of the use of institutional funds in a major mining project is the $148 million loan by a group of insurance companies to Reserve Mining Co., which is owned equally by Armco Steel and Republic Steel. Reserve is developing a large scale operation in mining and beneficiating magnetic taconite ore which it has leased from Mesabi Iron Co.

Prior to arranging this insurance company financing, Reserve had expended something like $8 million to establish an initial operation to yield 300,000 tons of pelletized furnace feed (60 to 65 pct iron). The loan is expected to supply all the outside capital required to achieve the next objective of 2½ million tons of annual output. In moving to the third stage of 3¾ million tons per annum the controlling companies might supply part or all of the additional $10 to $20 million of capital requirements. At that point, the total capital investment would be in the order of $170 million and to achieve the ultimate goal of 10 million tons of output per annum might require an additional $130 million of capital.

Another example of institutional participation in a major mining project is the Sherritt Gordon development of its Lynn Lake nickel-copper-cobalt property in Manitoba.

Government Agencies

Government aid and participation in mine financing has expanded tremendously over the past two decades, having received its first impetus during the depression of 1929-32. Further stimulus came during the emergency period of World War II and from the post-Korea defense program.

It is the authors' opinion that while the various agencies handling government financial aid may have been initiated under legislation to meet temporary situations, the general acceptance of government aid continues to increase in scope. More recently, the Paley Report recommended continued government aid to such projects. Also, more and more established mining companies are taking advantage of government financial aid.

It is not the purpose of this article to critically examine all of the aspects of government financing but to fit it into the scheme of financing of domestic mines. The authors wish to point out that government financing in general has been specifically designed as a public emergency aid and, thus:

1. A vendor is enabled to get financial aid from the U. S. Treasury under conditions which would not be allowed to public and private financial institutions under current Federal and State security laws.
2. As long as government continues to give financial aid and to take the ventures involving higher risks, less financial aid can be expected from private and public sources. In the long run such public financial aid might disappear.
3. Government not only can take the larger risks but can also pay for examination of the properties, and assembly and preparation of the necessary data for obtaining the financing.

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Government has had an increasing effect on mine financing—through taxation

4. Government loans are being made to bring into production mining properties of a marginal nature which probably would not be financed from public sources until a much later date. However, should any recession in prices of metals occur, such marginal properties might require subsidy or be required to shut down.

The principal government agencies involved in mine financing are the DMEA, the DMPA, and the RFC. These agencies afford help either by direct or indirect financial assistance, or by direct loans. Such aid is obtainable for ventures in the explorative stage (DMEA), the intermediate development to production stage (DMPA), and the final major financing to productive stage (RFC).

The DMEA

The Defense Minerals Exploration Administration was set up to carry on the minerals exploration assistance program authorized by the Defense Production Act of 1950, as amended, for the purpose of encouraging discovery and development in the U. S. and its territories of additional deposits of critical metals and minerals vital to national security.

The 34 minerals which the government seeks to have discovered and developed are divided into three groups. The 12 in Group A are the common nonferrous metals such as copper, lead, and zinc and for projects in this classification the government will supply 50 pct of the exploration costs. Group B consists of 6 items including antimony, manganese, tungsten, and rutile. In this category the government can supply 75 pct of the exploration costs. There are 16 items in Group C, among which are beryl, cobalt, columbium-tantalum, nickel, tin, and uranium. Here the government can supply 90 pct of exploration costs.

Indicative of the important role that the government is taking in aid to exploration projects through DMEA are the figures, released by this agency for contracts executed through November 1952. The 421 contracts have a total value of $20,286,288 of which the government's participation comes to $12,-242,390. Aid to exploration is included for 21 different mineral commodities, of which copper, lead, zinc, mica, and tungsten drew the major attention. The average value of the contracts is about $48,000. Featuring the willingness of the government to take a chance is the statistical record of DMEA. Out of 1607 applications for exploration assistance received during the above period, 421 or 26 pct were approved, 678 or 42 pct were denied. The balance, 32 pct, had been withdrawn or were still in process.

There is little doubt that any private company, regardless how large its resources, would hesitate to employ its funds so hopefully on exploration projects. Of the 1607 applications offered DMEA, it seems safe to say that at one time or other most of them had sought financial aid from mining and exploration companies before they approached the government.

The DMPA

The Defense Materials Procurement Administration assistance program is more comprehensive than that of DMEA. The basic job of DMPA is to stimulate new production and when possible to utilize private capital and industry cooperation.

The methods used by DMPA to stimulate production and procurement include outright purchase, floor-price guarantees, loans, certificates of necessity, assistance in procurement of equipment, and subsidies.

DMPA may contract with a mining company for purchase of a mineral at a specific rate. Such contracts are usually negotiated at a price below current free market, although sometimes above domestic ceilings, but at a sufficient level and for a sufficient production to take most of the risk out of the project.

DMPA may subsidize a mining company directly or buy products at over ceiling price to keep production going if operating costs are too high to show a profit.

DMPA, contrasted with DMEA, does not take many risks. Its emphasis is on procurement and a contract is only made after careful investigation as to probability of performance.

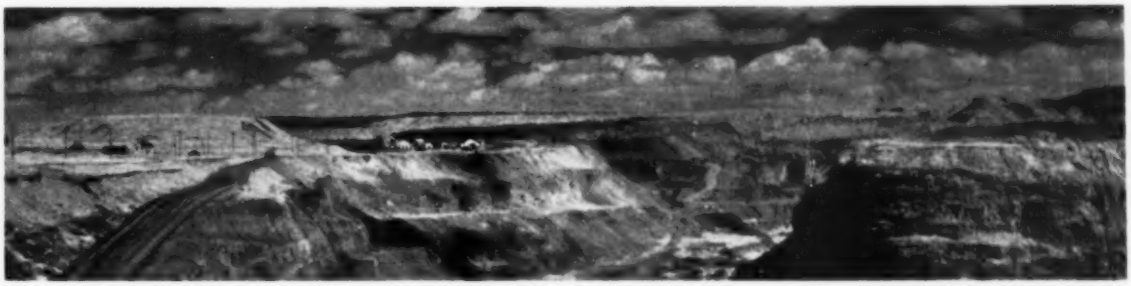
Each case handled by DMPA must be realistic as regards amount of production, time, and size and amount of facilities available. The goal is production which can be counted on within a fixed period, not assistance in mining activities.

The RFC

Under procedure up to May 1953, if government loans were needed to get into production, an application was made to Reconstruction Finance Corporation, which may act on its own initiative utilizing its own funds, since RFC may make loans for mining purposes on its own statutory authority without reference to any defense activity or program. In 1938 RFC was empowered to buy securities from business. Borrowers were eligible when credit was not otherwise available. Finally in 1945 RFC was given power to guarantee loans.

In granting a loan, RFC requires a substantial subordinate financial participation by the borrower. No loan is made on hopes only; the project must have demonstrable substance and feasibility.

Essential differences in the work of the respective agencies may be summarized as follows: DMEA par-



—through regulation of securities—and most recently in the role of financier

ticipates in financing only for *exploration* purposes. It will issue a certificate of discovery upon location of a substantial and probably commercial orebody as a result of an exploration program. DMPA has a *very limited* program for aiding in development between exploration and production, but the financing is primarily to aid actual production rather than development. The policy is to induce private capital to take over the development phases of mining operations. RFC has no program to finance mining prospects prior to the production stage.

The White Pine Project

The White Pine project of the Copper Range Co. may be cited as an example of substantial government assistance in the development of a major mining property. In this case, as in other similar ones, the government assistance has three principal component parts: 1—long term loan of capital funds, 2—certification of a portion of the cost for accelerated depreciation, and 3—a government procurement contract. The White Pine property comprises 9840 acres of fee, mineral and surface rights located in the Upper Peninsula of Michigan and owned by White Pine Copper Co., which is a wholly-owned subsidiary of the Copper Range Co. The property contains more than 300 million tons of proven and probable ore averaging about 22 lb of copper per ton. The project calls for construction of all facilities for mining, milling, and smelting (also townsite, etc.), for the production of 75 million lb per annum of high grade copper. It is planned to bring the property into production by the end of 1954.

RFC, acting as agent, authorized a loan to White Pine Copper Co. of $57.2 million. This was the first loan approved in this particular manner involving both RFC and DMPA, and the procedure was worked out because the project did not qualify for direct RFC action under the powers derived from its own enabling legislation. It is understood that the loan application had received the endorsement of the DMPA, the Munitions Board, the DPA, and several other agencies. The time required for negotiation of this loan was much longer than is usual for smaller and less complex projects and far exceeded the 3 to 6 months which is normal for DMEA projects. Negotiations were initiated in September, 1950, and the authorization was granted on November 15, 1951.

The RFC loan matures in 20 years, carries a 5 pct interest rate and is secured by first mortgage on all of White Pine's fixed assets, existing and to be constructed. It has a fixed sinking fund of $3 million per annum, payable quarterly, which becomes operative July 1, 1956, or 18 months after completion of the project, whichever date is earlier. There is also a contingent sinking fund requirement, designed to assure that a minimum of 50 pct of the operating profit of White Pine (after taxes but before bookkeeping charges) shall be devoted to debt retirement. It is further stipulated that any tax savings resulting from accelerated depreciation shall be applied against debt retirement installments in reverse order.

The RFC loan does not provide funds for working capital or for interest during construction. The Copper Range Co., the parent, has agreed to provide funds for these purposes, estimated to require advances to the subsidiary of about $13 million over a period of seven years.

On Nov. 16, 1951, the DPA certified to the Commissioner of Internal Revenue that the project was in the interest of national defense. Facilities estimated to cost $42 million qualify for accelerated depreciation, applicable to $29 million (70 pct), which may be taken during the five calendar years following completion of the project.

White Pine's procurement contract was entered into on February 25, 1952, and provides that DMPA will purchase at 25.5¢ per lb up to 243,750 tons of the first 275,000 tons of copper produced, if requested by the company. The government retains an option to purchase all of the 275,000 tons under the contract at the 25.5¢ price or the then current market, whichever is higher. The 25.5¢ price is subject to adjustment according to changes in government labor and commodity indices, and the contract terminates on December 31, 1961, or at an earlier date under certain contingencies.

It is readily apparent from this brief summary of the White Pine project that the government assistance program provides capital for major mining projects on a basis with which no private or public financing agency or medium can compete. In the absence of such government assistance, this project might have been delayed an indeterminate number of years, and when undertaken would undoubtedly have been developed on a much smaller initial scale.

Established Mining Concerns

Up to World War II, the mining companies played an important part in supplying financial aid for new mining ventures. In recent years, because the maturity of our prospecting status and the greatly increasing part taken in financing mine ventures by the government, the mining companies are going into scientific prospecting to maintain a flow of new ore supplies for future requirements.

The established mining companies exhibit considerable diversity in their approach and proposals for extending financial aid to projects brought to

their attention. However, there are three basic patterns which emerge from a study of the entire field.

Option to Purchase: This procedure is used where it is possible to set a price in relation to an appraised value and the amount of investment is not very large. Payment may be in a lump sum if the amount involved is not large. However, payments are usually made over a period of years in predetermined sums. Such sums usually bear a close relationship to the estimated amount of money that the property could afford to pay out as an annual royalty when brought into production, and such payments are applied against the price set in the original agreement. The basis of royalty calculations may vary from 2 to 15 pct depending upon the average value of the ore, and are generally applied to the net smelter returns. Ten pct may be considered a good royalty. For numerous nonmetallic minerals of low value the royalty rate is frequently below 10 pct. In many instances the mining company agrees to a more or less continuous program of development to the production stage.

Lease on Royalty Basis: This method applies in cases where it is not possible to fix a purchase price, and it is not generally attractive to the mining company because the royalty may become burdensome on operating costs. A sliding scale royalty may be preferred in such cases by the mining company. It also prefers to have complete control over the property, especially if there is considerable investment required in surface and underground facilities.

Participation: This plan has considerable variations depending upon the financial requirements and reliability of the vendor. The company would insist upon full management control and in most cases it prefers to make provisions eventually to purchase the equity retained by vendor. Provisions are made for the company to recoup its capital investment prior to the division of profits. The equity retained by the vendor may vary from 25 to 50 pct in case the venture is a moderate sized one. In the case where the financial requirements are large, the equity may run from 5 to 15 pct or more. The disadvantage of this plan is the potential interference in managerial prerogatives from minority interests.

A practical modification of the participation plan is that of offering the vendor a reasonably good earning rate without having to participate in cost of bringing property to production. The vendor may be given something in the order of 4 pct of the profits after amortization but before federal income tax.

Before an established company will initiate any financial aid it will require well organized information and engineering reports on the property. Mining staff members are very adept at spotting so-called reports written by amateurs, and such reports prejudice their consideration. At this stage, the company may wish to discuss the broad outline of a deal. It is unusual for any payment to be made to the vendor until the mining company has had ample time to examine the property and substantiate some of the claims of the vendor. In cases where the vendor has spent a considerable sum in an initial investment he is offered a refund or "standby" funds. The vendor might also insist on a reasonable amount of development work in bringing the project to production. In cases where outright purchase is contemplated, a period of two years might be required before final negotiations can be terminated, because of the development work necessary to insure an adequate return.

An established mining concern can offer certain advantages to a vendor. It is likely to have the knowledge and staff to obtain early development results and solve difficult technical problems. It is usually in a position to market the product to greatest advantage.

There are limitations to the extent of financing that it is possible to secure directly from a mining concern. Where several millions of dollars are involved, the mining company might find it necessary to seek supplementary funds from outside sources.

It is obvious from the methods used by mining companies in financing properties, that there is room for trading to the advantage of the vendor, provided his data have been well organized and substantiated. It is also obvious that in presenting a venture to a mining company, the vendor can expect to relinquish control and management of the property.

The Vulcan Project

Although the contract between Vulcan Silver-Lead Corp. and American Smelting & Refining Co. for the exploration and development of the former's property is not typical of the usual situation involving an individual vendor and an established mining concern, it does have a number of features illustrative of some of the attitudes and procedures.

The old Galena property is located in the Silver Belt of the Coeur d'Alene district, about 2 miles west of Wallace, Idaho, and some 5 miles east of the properties of Bunker Hill and Sullivan and Sunshine Mining. In 1946, Callahan Zinc-Lead Co., owner of this property, organized the Vulcan Silver-Lead Corp. and transferred to it roughly the western two-thirds of the Old Galena property, including the 800-ft deep Galena shaft.

The basis for the deal with American Smelting was the geological inference that commercial ore existed at 3000 ft depth (sea-level).

In 1947, Vulcan Silver-Lead signed a 30 year lease agreement with American Smelting & Refining Co., with a 30 year extension privilege. AS&R undertook exploration and development work at its own expense, involving sinking the existing shaft to the 3000 ft level, estimated to cost up to $1 million. Vulcan was to receive $500 per month as advance royalty, recoverable by the lessee out of future earned royalties. With the production of ore, Vulcan would receive a royalty of 10 pct of net smelter returns, subject to reduction for unrecovered advance royalties. After the lessee, (AS&R) should have recovered its capital costs and accumulated a working capital fund of $500,000, then Vulcan would be entitled to 50 pct of all profits.

By the middle of 1950, after 3½ years during which nearly $1 million had been devoted to the project, the shaft was bottomed at a depth of 2930 ft, and an exploration level driven. The encouraging ore showings encountered by the end of 1950 resulted in the suspension of exploration except for diamond drilling while the shaft was rehabilitated and improved, to enable handling 1000 tons a day as compared with the former rating of 150 tons a day. This phase of the project occupied the time from the end of January 1951 to May 1952, and since that time underground development work has been resumed.

By the end of 1951, American Smelting & Refining had raised its total expenditures to about $1.9 million and at the end of 1952 the total had exceeded $2.5 million.

Among the interesting collateral aspects of the project is the fact that Day Mines has a 25 pct participation in AS&R's side of the contract and that Federal Mining & Smelting Co., an AS&R subsidiary (now absorbed into the parent company) had a 15 pct participation. On the other side, Callahan Zinc-Lead, since 1946, has disposed of a number of blocks of its stockholding in Vulcan Silver-Lead so that its present ownership is about 56 pct.

Another interesting illustrative project, involving a production program rather than prospecting or exploration, was the operating contract between Benguet Consolidated Mining Co. and Consolidated Mines Inc. Both of these companies are American owned, although they have headquarters in the Philippines, and the principles involved in their contract are applicable in this country.

Benguet is one of the important gold mining companies of the world with a long record of profits and dividends. Consolidated Mines was short on capital, short on management experience, needed technical advice and marketing facilities in order to function profitably. Its sole asset was the ownership of the 10-million ton low-grade Masinloc chromite deposit.

Under the terms of the contract, Consolidated agreed to maintain and safeguard its rights in its claims and Benguet agreed to explore, develop, mine, concentrate, and market the ore found on the claims. Benguet agreed to provide funds from its own resources as necessary for exploration and development, plant construction, and for transportation until the properties show profit.

Benguet shall be reimbursed out of actual net profits from operations of Consolidated to the extent of 90 pct of such net profits. The remaining 10 pct is to go to Consolidated for expenditures made out of its own funds prior to the time the properties are put on a profit producing basis. Expenditures from Benguet's own resources shall be charged against the subsequent gross income. After Benguet has been fully reimbursed for its expenditures, net profits from the operation shall be divided evenly between Benguet and Consolidated.

Benguet has the right on 90 days notice to cancel and terminate the contract. If upon termination, the company has not been fully reimbursed for equipment and installations it has made on the property, Benguet shall have the right to select equipment for removal to reimburse itself.

This contract has been exceptionally profitable to both parties and present shipments are being made at the rate of over 50,000 tons a month.

Public Financing

If it is proposed to raise funds for mine development through a public offering of shares, registration of the issue is required with the Securities & Exchange Commission as well as compliance with the provisions of the various State "Blue Sky" Commissions. In most states the requirements of these Commissions do not involve a great deal of work or expense. For example, in Massachusetts, it will suffice to file with the Commission a notice of intent to sell by a registered broker or salesman, together with a simple form revealing general corporate details, or in its stead, a copy of the registration statement as filed for the security with SEC. The Commission may investigate any security when the information furnished is believed to be inadequate.

The requirements vary for the different state commissions and some are far more onerous than others. Registration of the stock offering with SEC does not in itself act as a clearance with the state commissions, but undoubtedly carries a lot of weight.

Registration with SEC is usually an expensive experience and has caused protest from promoters who blame the law for the decline in speculative offerings. Unfortunately, it is true that a thoroughly honest effort to obtain venture capital for a new mining enterprise must encounter the same searching investigation that is faced by a promotion of the most questionable character.

For issues of $300,000 or less SEC permits use of the so-called "Short Form" which does not require filing a registration statement with SEC, but merely sending a Letter of Notification, under Regulation A, Form S-36-1.

Although compilation of the letter of notification can be accomplished easily enough and filing of a prospectus is not a requirement under Regulation A, it is required that any sales literature that the underwriter may wish to use must be filed and cleared with SEC. As a practical matter it is virtually impossible to obtain funds from the public with only the letter of notification and its abbreviated statements and information. Something more is needed to excite the investor's interest and to arouse the enthusiasm of the speculator. Thus a prospectus, or offering circular as it is usually called when used under Regulation A, is a "must" in any sales campaign and in the preparation of such literature the same thoroughness and care must be exercised as in preparing for full registration and prospectus under the Long Form, S-11; for this material may, and probably will, be questioned just as sharply and fully as under S-11.

Form S-11 for registration under the Securities Act, when used for shares of exploratory mining companies, calls for considerable care and expense in its preparation. In addition to the cost of preparation of the registration statement, there is the actual cost of printing the statement, making amendments which aside from the delay may sizably increase the printing bill, and printing the prospectus. Perhaps $15,000 to $20,000 may approximate the total, but higher figures have been quoted.

It has been pointed out by an officer with SEC that by far the greater portion of the cost of floating a security issue with the public, particularly in the case of small issues, consists of underwriters' commissions and discounts and not the actual expenses of registration. Presenting figures for the year 1945 through 1947, the SEC Chairman contended that the average cost of flotation for nineteen registered issues of $500,000 or less was 21.9 pct, with the underwriters' commissions and discounts averaging 17.7 pct, or about 80 pct of the total flotation costs. tion costs.

It appears that this statement is quite in line with the facts but it should be noted that the spread can be considerably wider in the case of more highly speculative offerings where the amount involved is less than $300,000. A typical instance of this sort was the "ABC" Tungsten Mining Company for which the total public offering of stock amounted to $299,850. Underwriting discounts and commissions

were estimated at $74,962, or about 25 pct of the total price to the public. Expenses of the issue, including legal and accounting expense in connection with the underwriting, totaled some $25,000 or about 8 pct. Thus, there was a total spread of 34 pct of the price to the public or a mark-up of close to 50 pct above the net proceeds to the company.

In summary, the time lost in preparing a registration statement and the expense involved in obtaining the detailed, accurate information required by the Commission, have discouraged the mining claim-owner from seeking development funds through public financing.

Add to this the high "take" of the underwriter and the vendor is often under considerable pressure and temptation to underestimate his financial requirements, particularly if by so doing he can keep his offering below the $300,000 total and thus use the Letter of Notification under Regulation A instead of full registration. Thus, in many cases the new venture finds itself short of working capital and the embryo mine development dies aborning.

The public financing picture is indeed rather discouraging to the seeker of venture capital for small scale mine financing. So widespread is the belief that a stock offering pursued through legal channels is a thoroughly unsatisfactory procedure that for many years few registrations have been entered with SEC for mine financing using the "Long Form." Those using the Letter of Notification, or "Short Form," have been usually of ultra-speculative character.

SEC has published a list of certain views bearing on mine financing that are excerpts from various decisions following "stop-order" actions. Space does not permit a complete recital of the whole list, but some of the more interesting views may be commented upon.

A sharp distinction is drawn between the development and productive stages of a mine and it is held to be misleading to imply that a property is in production when the work is only exploratory or development in character, even if ore is being milled or shipped. In reverse, it is misleading to claim a property is in the development stage when it is actually in production, particularly if operating losses were capitalized to conceal the character of the work.

Especially emphatic is SEC'S insistence that proper engineering procedure must be used in describing ore reserves. When data are presented relative thereto, there must be present a systematic collection of facts regarding the ore bodies, or in plain words, a conventional assay map compiled according to good engineering standards. Quantitative statements regarding tonnages of "possible ore" are held to be misleading, hence ore reserves can properly include only positive and probable ore, ore in the approved nomenclature, measured and indicated.

It is held to be misleading to state the average value of a number of samples based on a calculation which took no account of the sample width of the respective samples. When property valuation is based upon estimated earnings from mining operations to extend over a period of years, the expected profits should be reduced to present value using the Hoskold formula, or some similar method.

None of the opinions of SEC will conflict with professional standards used by experienced mining engineers and should entail no difficulties in following them. However, they do make it hazardous for a promoter without technical or practical experience to attempt to write a prospectus that will be really informative, yet enthuse purchasers. The commission has emphasized that it is not only necessary to disclose facts, but it may be misleading not to reveal something the suppression of which may give a wrong impression of the merit of the property.

Of more importance to the promoter is the requirement of complete disclosure of the real cost of the property to him, prohibition of a pro-forma balance sheet unless a firm commitment has been made by a responsible underwriter, disclosure of discounts and commissions to underwriters, net amount that will go to the company after payment of the underwriting spread, and details of the application of the proceeds. Obviously, the more the promoter and the underwriter get out of the financing, the less goes into the company's treasury and in cold figures this may make a rather discouraging picture to the prospective investor.

The reputation of the underwriters and the promoter are all-important in the success of public financing. A well-known, thoroughly reliable investment house can undertake the financing of a mining property and make disclosures in the prospectus that would appear to frighten away even the most reckless speculator, yet investors may be eager to subscribe for shares because they rely on the integrity and sound business judgment of the firm that offers them.

Summary and Conclusion

By way of recapitulation and summarization of the subject matter of this study and viewing it in broad prospective, it appears to the authors that the financing of new mining ventures in this country, particularly if they are medium or small sized projects, is beset with many problems quite apart from the heavy expenses which may be involved, and that many of these difficulties are the result of long range trends which have been exerting greater impact upon the development of the mining industry and which are largely beyond the control of the individuals involved.

In addition to this, and inescapably involved, there is the growing importance of government participation in the field of mine exploration, development, and financing. This trend, which had been in progress at a slow pace over a long period of years, has been markedly accelerated over the past two decades. The increasing tide of government control and intervention has not been confined to the extractive industries alone, but it is readily understandable how this trend is particularly pronounced in the field of mineral resources. It may be rationalized, if not fully and completely justified, from the standpoint of the government with respect to national welfare and security. Moreover, it may be pointed out that this invasion of the spheres of private initiative and venture tends to be enlarged by other basic policies of government, such as strict regulatory policies and taxation of corporate and personal incomes.

Reprints of "Financing Domestic Mining Ventures", including Parts I and II, are available at a cost of (35¢) per copy. Due to cost of handling, remittances must accompany orders for reprints of this article.

Coal Burning Boiler Plants Can Be Economical for Small Mines

by D. M. Given and C. A. Marshall

HOW much can the mine operator afford to spend on his coal burning boiler plant? The usual answer is as little as possible and still get the job done. Thus, attention must be focused on the spot where excessive costs are most likely to arise — the materials handling end of the job. Boiler plants consuming less than 30,000 tons of coal annually can be controlled to the point where costs for handling coal and ashes are under \$1.00 per ton. There is a wide selection of coal and ash-handling systems that can be used in smaller boiler plants to provide labor savings and convenience required if management is to accept coal as fuel.

Economy of fixed charges as well as operating costs must be considered in selecting a system. In balancing fixed charges against operating costs, to evaluate different systems, all handling charges should be considered on a common basis of cost per ton of coal consumption. Table I illustrates labor charges per ton for full time coal and ash men in various sized plants, based on a \$3600 annual wage. It also shows that the limit of 25¢ per ton will not support full time handling labor in plants burning less than 15,000 tons of coal per year. In this smaller type of plant it is essential to avoid job classifications which cannot be assigned to the boiler operator or part time labor.

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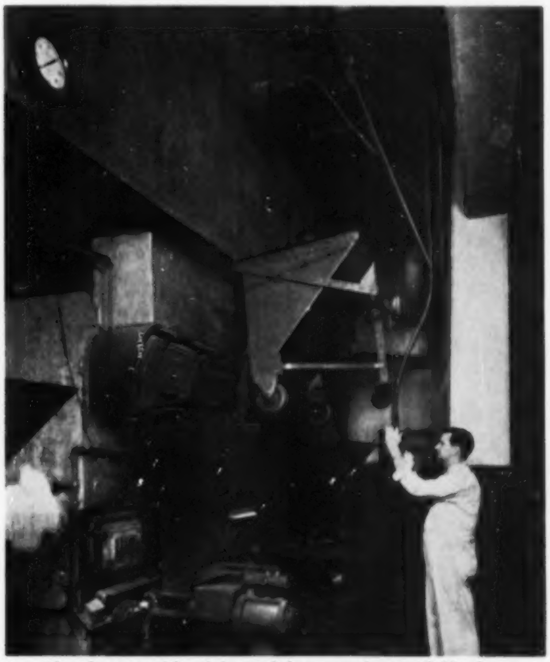
Maintenance plays an important part in costs. Because investment limitations are a factor, it is not always possible to select the handling equipment giving the lowest maintenance cost. In the plant category considered in this article maintenance costs may run from 15 to 25¢ per ton of coal consumption. Improperly designed systems can shoot the cost even higher.

The graph on the next page presents costs of systems considered adequate for several plant sizes. It also indicates that fixed charges can be controlled to about 45 or 50¢ per ton of consumption, using a conservative fixed charge rate of 15 pct per year.

A limit of \$500 to \$5000 is indicated (p. 686) for plants burning less than 5000 tons of coal annually. Off-track plants receiving truck deliveries directly to the storage bin need mechanization only from storage bin to stoker. Unless the siding is located directly over the storage bin, rail deliveries require additional equipment. A portable belt conveyor costing between \$300 and \$1500, inserted into the shallow pit below the tracks will serve to move coal to storage areas. An additional investment of \$1000 may be justified for a portable car unloader which eliminates need for pitting below car tracks. The



LEFT: Lift truck equipped with shovel attachment scoops coal into stoker hopper with minimum labor, remains available for other jobs. RIGHT: Monorail hoist with coal and ash bucket is another cost-cutting installation to reduce labor force needed in boiler plant.

Coal Used, Tons Per Year	Labor Cost, Dollars Per Ton 1 Man	2 Men	3 Men
1,500	2.40	4.80	7.20
6,000	0.60	1.20	1.80
10,000	0.36	0.72	1.08
15,000	0.24	0.48	0.72
20,000	0.18	0.36	0.54
30,000	0.12	0.24	0.36

Labor Cost Per Ton of Coal at $3600 Annual Wage Rate

unit rests on top of the tracks and moves coal from under the car to the portable conveyor.

Installed cost of a simple monorail hoist runs from $500 to $1000. The boiler operator fills the wheel-mounted bucket with coal and pushes it to the stoker hoppers. It is then hoisted and pushed along a monorail track for dumping directly to the stoker hoppers. Bucket capacities to 700 lb are practical.

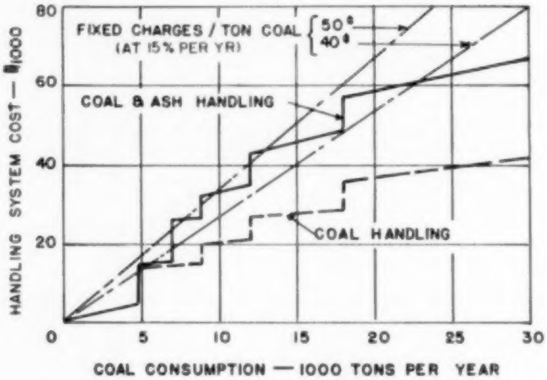
Lift trucks may be equipped with a scoop attachment for handling both coal and ashes. Prices range from $2800 to $4500, depending on capacity and tires. Bucket capacities from 750 to 2000 lb are practical.

Bin-fed stokers are used successfully in many smaller plants. The stoker feed screw is simply extended back to the storage bin. With hopper-model stokers, a separate screw conveyor can be installed between the storage bin and the stoker hopper. Installation price runs between $700 and $1500.

Silo System for 15,000 Ton-Per-Year Plant

A plant with a maximum steam load of 85,000 lb per hr and a 40 pct load factor will consume about 15,000 tons per year, or an average of 300 tons per week. A concrete-stave silo with 300 ton capacity, of which 20 pct is on a live shelf, would cost $7500 to build. Unloading conveyors sized for 35 tons per hr would permit one-shift handling of more than four times the average daily consumption. Track hopper, feeder and bucket elevator would cost approximately $12,000 installed. The screw conveyor would be sized for about 8 or 9 tons per hr at a cost of $3000 to $4000 installed.

With the addition of $3000 for automatic scales and $2000 for car shaker and car thawer, total cost of a system handling about 15,000 tons per year is approximately $28,000. Modifications can either reduce or increase the cost. For example, a skip hoist



Typical installed costs of well-designed handling systems.

instead of a bucket elevator would raise costs another $20,000.

Arching Troubles

Some equipment designers have given little attention to the arching of wet coal in storage bins and chutes, beyond provision of "poke holes" and vibrators which the boiler operator may resort to in emergencies. The excuse that coal is too wet is not valid. Handling systems should be designed for coals that might be used, rather than coals economically unavailable to the plant. Any type of bin or chute can be designed to avoid arching troubles. Basically, the following must be considered:

1. Discharge openings from a bunker or silo should have a minimum diameter of 14 to 18 in., depending on maximum moisture content of the coal. Conveyors or feeders from the bin discharge should be equally as wide because the minimum cross section prevails in any gravity-flow system.
2. Storage shelves and bin bottoms should be sloped at 45° to avoid spontaneous combustion. Steeper slopes are to be avoided because of the "wedging" effect created above the discharge, even if some coal with a steeper angle of repose has to be hand trimmed off the slope in emptying the bin.
3. When practical, chutes and bins should be designed for an angle of repose of about 60°. Thus, a silo with a height to diameter ratio of about 3 to 1 offers substantial advantage over shallow overhead bunkers some of which may have less than 50 pct of capacity in true live storage.

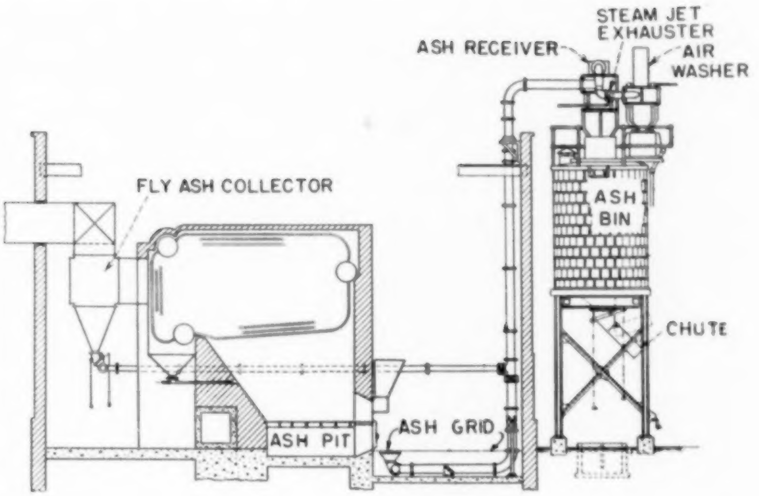
Ash Handling

Coal handling convenience should take preference over ash handling. Hoist buckets and other portable

Typical Installed Costs of Handling Systems

Coal Used, Tons Per Year	Type of Handling System Coal	Ash	Coal Silo, Tons	Track to Silo, Tons Per Hr	Installed Cost, Dollars Coal		Ash	Investment, $ Per Annual Ton Coal
300	Monorail Hoist		......	......		800		2.67
2,000	or Bin Feeder		......	......	800	to	3,700	to 1.85
5,000	or Lift Truck		......	......	1,200	to	5,000	to 1.00
6,000	Silo[1]	Hoist	120	25	14,000		1,000	2.50
9,000	Silo[2]	Pneumatic	180	25	20,000		12,000	3.71
12,000	Silo[2]	Pneumatic	240	25	27,000		16,000	3.58
18,000	Silo[2]	Pneumatic	360	35	36,000		21,000	3.17
30,000	Silo[2]	Pneumatic	600	50	42,000		25,000	2.23

[1] Portable conveyors only from track hopper to elevator
[2] Includes track hopper and car shaker

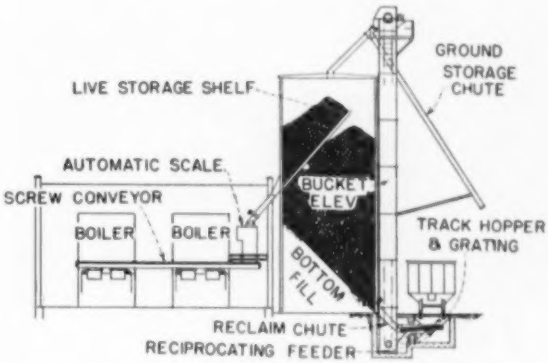


LEFT: Pneumatic system can be handled by boiler operator, except for an extra man in basement during ash handling periods. Ash intakes are in the operating floor directly in front of each ash pit door.

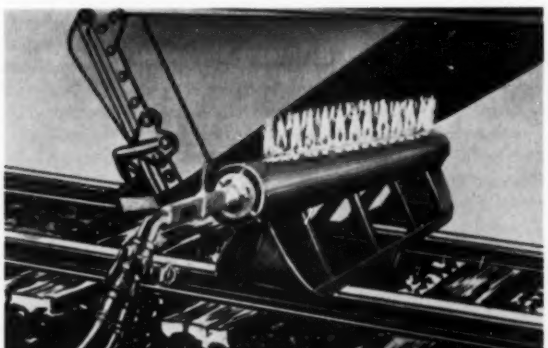
systems are ideal for small plants. Pneumatic or hydraulic sluicing systems for ash disposal start at $12,000 erected. Where ash cannot be sluiced to a fill area, decanting tanks are required, making sluicing systems excessive in cost for the plants considered here.

Figure above shows a pneumatic system with intakes in the operating floor directly in front of each ash pit door, and fly intakes at each fly-ash collector. The boiler operator can operate this system, including raking of bottom ash to the intake openings. These intakes may be located inside the ash pit or in the basement under a conventional ash hopper. Provision should be made for easily extracting and breaking up large clinkers. An extra man would have to be available for basement work during ash-removal periods.

Disposal system air flow is produced by a steam-jet ejector arranged to prevent steam mixing with the ash and causing storage bin freezing. An ash receiver is placed ahead of the steam jet to discharge the ash to the storage bin. Exhaust from the steam jet is then piped either to an air washer or to the rack, minimizing the amount of ash discharged to the atmosphere.



Push-button system for handling coal uses bucket elevator to upper bin section, and gravity feed to boiler-feeding screw conveyor. System has minimum handling requirements and would be suitable for installation in an older, congested location.



Portable thawing equipment, ABOVE, and inexpensive trackside car shaker, RIGHT, raise efficiency and lower handling costs in areas with severe climate.

Lift Speeds Timber Handling

Hard pressed by rising costs and competition from other fuels, coal mine operators are coming up with plenty of cost-slashing ideas. Many methods used in coal mines provide more than the germ of an idea for noncoal miners—some devices would work even better in hard-rock mining. Timberlift, displayed at the recent Coal Show, is one of these devices that may see wider usage.

Engineers of the Lansford, Pa., operation of the Lehigh Navigation Coal Co. called on Ruger Equipment Inc., of Uhrichsville, Ohio, to develop a device to make each timber handling crew a mechanized unit. Timberlift is the result.

Designed to provide each timber crew with inexpensive mechanization, independent of power sources, the lift picks timbers up from tunnel floor or flat car, raises, and holds them in place. Low headroom requirements lift capacity greater than 8 ft., and hand-powered hydraulic lift cylinder permit flexible use.

Lift swings timber from car onto cradle before pushing it up into place.

Timber is lifted from cradle and held in place.

Cradles built for timber 6-in. size and larger are vital part of the operation, enabling the arm to first pick timber up with a chain, rest it on a cradle, then push it into place. Only 20 to 40 lb of effort is required on the hydraulic pump, and the operation of raising and placing timber takes less than a minute.

With a 2000-lb capacity, Timberlift weighs about 350 lb, and only mounting required is for swivel base permitting 360° rotation of lifting arm. Present price is approximately $500 complete, ready to install.

Hardening for Increased Bit Life

Special interest attaches to any means of increasing resistance to wear and abrasion, for in addition to mechanical wear common to all industries, mining has additional problems of bit and tool wear in handling its basic material: rock. One new method being tested is bit hardening with a product sold under the name of Hard-N-Tuff. Claimed to offer major improvement in wear resistance, it is a steel hardening compound in powder form.

Three hardening actions, carburizing, nitriding, and chromizing are stated tc be accomplished in one operation. Photograph shows treated and untreated detachable bits after test. Churn drill bits operating in taconite were reported to show 25 to 80 pct increase in bit life for hardened bits over standard ones.

Regular heat treating facilities, muffle furnace, gas flame, open forge, or induction heating, can be used for four application steps, heating, coating, reheating, and quenching. Treatment time is stated as less than five minutes, and may be carried out during normal resharpening.

Photograph shows untreated bit at left that drilled 6 in. Bit at right was used as starter for 12 in.; drilled another 30 in. for total of 42 in., or a 5 to 1 improvement in wear. It can be resharpened. Tests were carried out in hard hematite, where standard carbide insert bits were stated to last only 36 in.

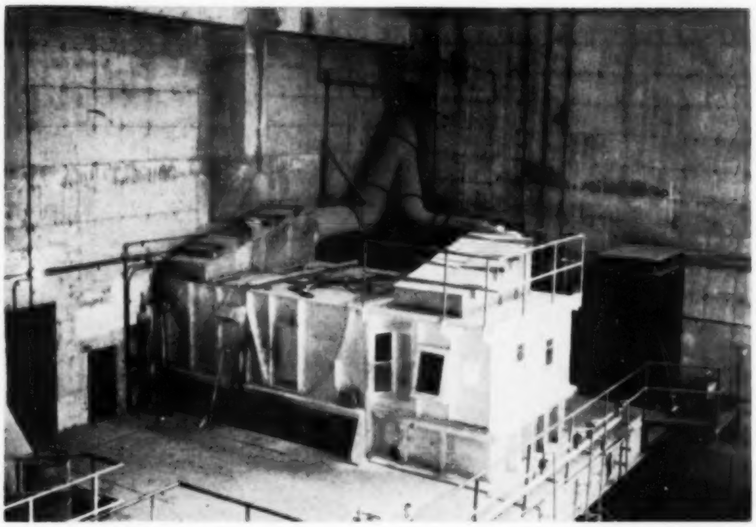


Fig. 1—Pan-feeder and jaw-crusher feed enclosure at the new Ray crushing plant. Photograph shows exhaust hoods and ducting.

Crushing Plant Dust Control at The Ray Mines Division, Kennecott Copper Corp.

by John F. Knudsen

Following the flowsheet from the primary crushing plant of Ray mines division through the final products sample mill, there are eight exhaust systems ranging in size from 12,000 to 30,000 cfm. Where possible, hood face velocities are kept around 500 ft per min, permitting branch velocities of 2500 ft per min. Tight-fitting enclosures are provided over all ore drops, crushing operations, and vibrating screens. All enclosures are exhausted either primarily or secondarily, except those around conveyor head-pulleys where the ejector effect of the falling ore creates sufficient indraft to abate dust dissemination.

IN 1947 an Industrial Hygiene Department was organized to represent and assist the four western mining divisions in industrial hygiene problems. Department headquarters are located at the Utah Copper Division, Magna Plant, Magna, Utah, with a branch office at the Chino Mines Division, Hurley, N. M. The headquarters staff includes a director, a department secretary, an industrial hygienist, a supervising ventilation engineer, and two ventilation designers. To be in closer association with work at the Southwest properties, a field engineer at the branch office conducts field studies which include ventilation problems at the New Mexico and Arizona properties, i.e., the Chino and Ray mines divisions.

Functions of the department are to discover and evaluate existing hazards and to develop protective measures, either by personal protective devices or by proper ventilation, depending on the nature of the contaminant. Other than dust counting, all air samples collected at the four properties are processed at the home office. If through plant surveys or by suggestion of management some type of ventilation or air conditioning is required, the problem is usually handled by the ventilation section of the headquarters department, where general arrangement or line drawings are prepared, showing hoods, ducting, and equipment. Some hood and special parts detailing is done at headquarters; otherwise, required detailing is done by the respective plant engineering departments.

Prior to the design of industrial ventilation, either new projects or the revamping of old ones, a thorough survey is made of existing conditions. In so far as possible weighted average exposures are determined and adequate ventilation provided on the basis of maximum allowable concentrations of the contaminant in question, as well as medical case histories, if these are available.

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The Transactions papers appearing in Mining Engineering during 1953 will subsequently appear in AIME Transactions Vol. 196, and may be permanently referenced to that volume.

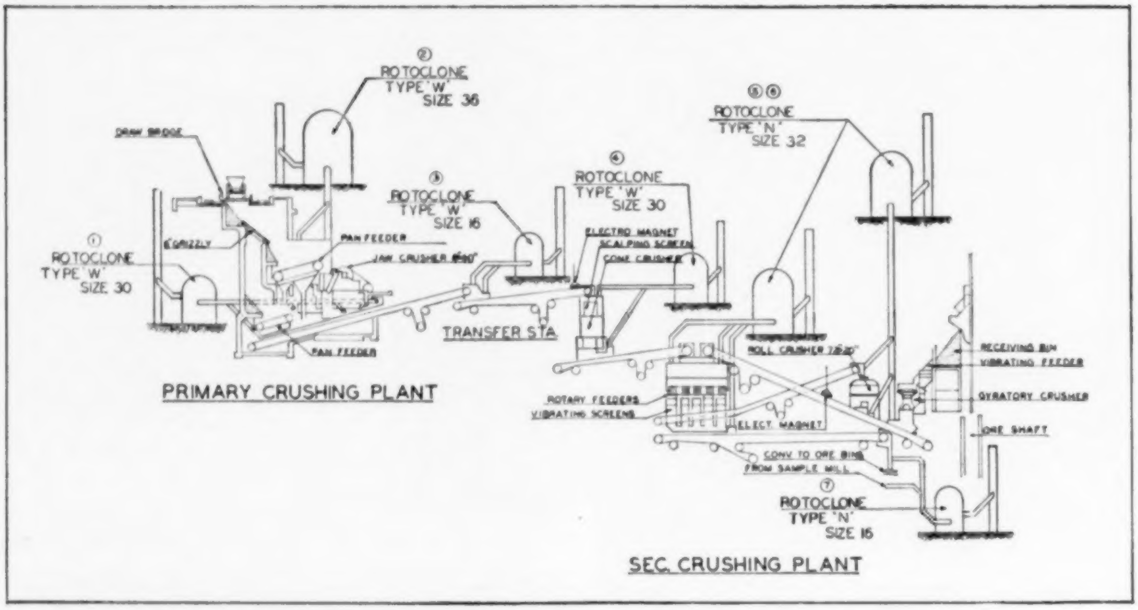

Fig. 2—Diagram of exhaust systems from primary crushing plant through final crushed product sample mill.

Until Kennecott's new Utah refinery came into the picture the major products of Kennecott Copper Corp. Western Mining Properties were copper concentrates and smelted copper; the principal concern of the Industrial Hygiene Department, therefore, was to control dusts in the mines and crushing plants. This was accomplished by dilution or by exhaust ventilation. In the crushing plants at Ray, exhaust ventilation was employed exclusively.

When the decision was made to augment the ore supply from underground with that of ore from open cut operations, it became evident that existing crushing plant facilities were inadequate to process the additional tonnage. As an underground operation the mine produced about 5000 tons of ore per day, whereas under the new plan the tonnage was to be stepped up to 15,000 tons per day, 12,000 tons from open pit operations and 3000 from the underground workings. The crushing is done on two 8-hr shifts at approximately 900 tons per hr.

Preliminary to the design of exhaust ventilation for the Ray property, rafter dust samples collected from the old crushing plant were analyzed and sent to dust collector manufacturers, who determined the settling rate, the particle size, and the collector efficiency ratings.

The main orebody consists principally of schist with intrusions of diabase. Chalcocite is found in the schist, intermingled with pyrite in ribbonlike stringers throughout the workings. Soluble copper, in varying degrees throughout the mine, is mainly in the form of copper sulphate. The percentage of soluble copper salts increased when operations in the open cut mine were begun. This created a problem of corrosion in the dust collectors, much more pronounced than was anticipated from knowledge gained by samples of rafter dust collected from the old crushing plant.

The schist type of ores were found to be water repellent, and when finally wetted became very adhesive, sticking to the conveyor belts and idlers. The dribble problem that ensued was most difficult to cope with because of the number and length of the conveyors. Obviously this characteristic ruled out the possibility of using any form of water spray preliminary to and during crushing operations.

Exhaust System Design Characteristics

Estimating of air volumes to be exhausted from enclosures and housings continues to be more of an art than a science, although several engineers in the field have made some progress toward standardization. In a paper published in 1949 an attempt was made, using water droplets, to determine the extent of displaced air caused by material falling into a tank.[1] The empirical formulas developed in this paper are not directly applicable for use in the field but were a start in the right direction, to be followed by a more practical plant operations approach.

On large-scale operations, where many variables exist, past experience and field experimentation constitute the principle criteria for establishing required exhaust volumes. Ideas of enclosing the

Fig. 3—Grizzly undersize product pan-feeder enclosure, showing access door and recessed lighting.

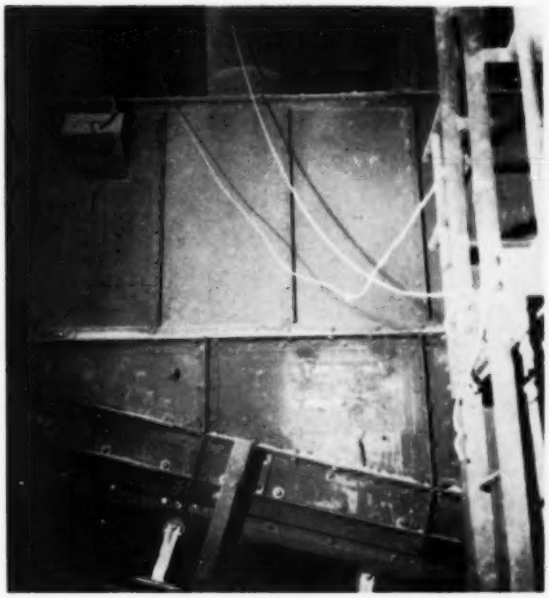

Fig. 4—Conveyor enclosure between crusher and pan-feeder discharges. Note distance of the exhaust point from the conveyor.

Fig. 5—Final take-off point on conveyor enclosure. The conveyor covers approximately 15 ft beyond the last dumping point.

many different types of machines and locating the exhaust hoods vary extensively, depending on the type of machine, the need for accessibility by the operators and maintenance crews, and the available space around the equipment for housing, hooding, and provision of adequately sized ducting. When exhaust ventilation is engineered conjointly with the design of the plant the pitfalls of inadequate space are overcome, but the situation is complicated by many unknown factors that cannot be tested before completion of the final installation. Some flexibility must be provided to cope with minor discrepancies in design.

Air flow distribution in exhaust systems can be accomplished in one of two ways, by orifices or dampers or by properly adjusted pipe sizes. Although employment of the first method is widespread, there are disadvantages such as rapid wear due to restricted area, tampering by unauthorized personnel, and often the complete removal of dampers. In so far as continued successful operation is concerned, the use of regulated pipe sizes is the better of the two methods. On Ray installations this method was employed on all dust control systems with the exception of the sample mill exhaust system, where diversified control was advantageous. With minor exceptions the system for balancing the air flow in the exhaust systems closely followed the method outlined by Drinker and Hatch.[2] Ducting loss calculations were based on a compilation of data taken from several sources[1-4] as well as field data developed by the ventilation department.

Wherever possible, hoods were designed so that entering face velocities could be kept under 500 ft per min. Otherwise the openings were located a suitable distance above the dust source to avoid inspiration of large particulate matter. Branch duct velocities could thus be kept in the neighborhood of 2500 ft per min. A good exhaust system is one maintaining sufficient indraft at the enclosure openings to prevent dust dissemination but transporting only floating dust through the piping system.

Exhaust System Installations

Opinions vary among the authorities in the field of dust control regarding the size of exhaust systems. However, it is the policy of this department to keep the installations small for greater flexibility. The design of the Ray primary and secondary crushing plants was so organized that the various crushing and screening processes were amenable to the smaller exhaust systems. Typical dust control installation is illustrated in Fig. 1.

Following the flowsheet, as shown in Fig. 2, from the primary crushing plant through the final crushed product sample mill, there are eight exhaust systems installed to date, with additional exhaust proposed for the underground ore-crushing operations. The systems range in size from 12,000 to 30,000 cfm.

Fig. 6—Primary screen enclosures, showing the hoods and the ducting.

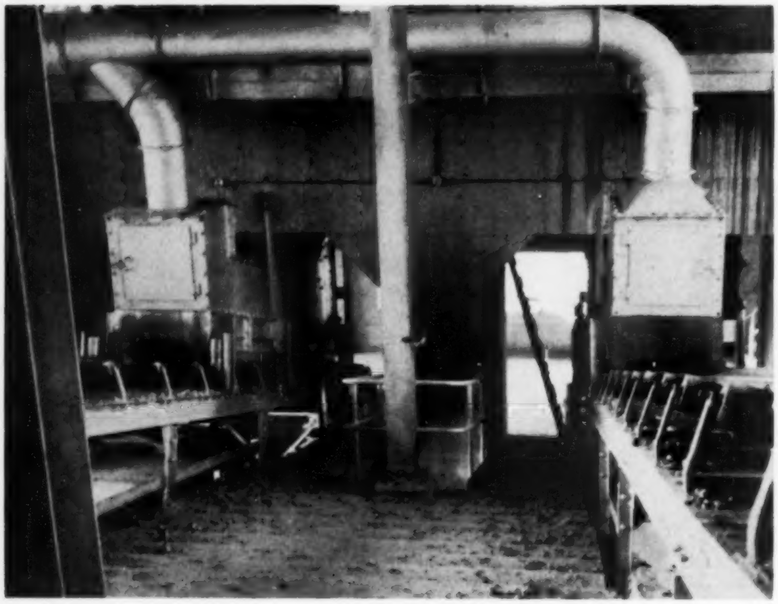

Fig. 7—Symons crusher area conveyor enclosures, exhaust points about 4 ft off conveyor. With this arrangement, a vertical riser precedes a horizontal run, reducing the possibility of material being projected into the branch ducting.

The ore from the open pit mine is transported to the crusher by 22-cu yd Euclid trucks. A two-gate horizontal slide portal admits the ore to the receiving hopper wherein grizzly bars separate the fines from the oversize product. The oversize material is carried by pan-feeder to the large jaw-crusher with the undersize, minus approximately 7-in. material passing to the undersize bin where it is fed to the No. 1 conveyor by a variable speed pan-feeder. Provisions were made for eventual exhaust from the receiving bin, if found necessary. However, secondary ventilation is provided by exhaust from the enclosure over the oversize pan-feeder and jaw-crusher and by ejector effect of fines passing through the grizzly section.

Table I. Primary Crushing Plant Exhaust Systems

Pan-Feeder, Jaw-Crusher Enclosure, System No. 1

Hood Location	Hoods, No.	Volume in CFM
Pan-feeder	1	22,500
Jaw-crusher feed	1	7,500
Total	2	30,000

No. 1 Conveyor Loading Point Enclosure, System No. 2

Hood Location	Hoods, No.	Volume in CFM
Tail of conveyor housing	1	2,500
Undersize feeder discharge	1	5,500
Crusher product feed	1	8,000
Head end conveyor housing	1	4,000
Total	4	20,000

Two ventilation systems, see Table I, are in operation over the 16-hr crushing period, one exhausting the enclosure over the pan-feeder and the jaw-crusher feed opening and one exhausting the feed points enclosure over the No. 1 conveyor. Fig. 1 shows the enclosure over the jaw-crusher feed opening and over the pan-feeder and jaw-crusher with exhaust hoods immediately in front of the portal through the curtain wall separating the receiving bin from that of the crushing plant proper.

As indicated in Figs. 3 and 4, the undersize material pan-feeder, as well as the discharge, is completely enclosed, with exhaust take-off points several feet above contact with the conveyor belt. All enclosures are well lighted, lights being housed in recessed receptacles to minimize breakage from flying rock. As illustrated in Fig. 1, operations are visible through windows provided on three sides. Access to the enclosure is provided by two sets of hand-lifted doors and one roll lift section, activated by air, for removing timber and other extraneous materials. Fig. 5 shows the final take-off point on the housing over the conveyor handling the final product leaving the crushing plant. The jaw-crusher discharges a product which is no larger than 10 in., one dimension.

Type W Rotoclones with sludge ejectors are used as a combined exhauster-collector; specifications are given in Table IV.

Transfer Station

The transfer station is furnished with exhaust ventilation both for protection of equipment and workmen, although men are not steadily employed at this location. The transfer point is completely housed; a type W Rotoclone, with exhaust hoods at the tail and head of the ore fall, exhausts 4000 cfm from the head and 1000 cfm from the tail of the enclosure.

Secondary Crushing Plant

As shown in the flowsheet, there are four exhaust systems in operation in the secondary crushing plant, with additions planned for the system exhausting the underground ore-receiving bins. When the system is completed, a volume of 99,300 cfm will be exhausted from the crushing and screening operations in the secondary crushing plant.

Symons Crushers Area: Ore from the primary crusher enters a distribution box immediately above the primary or scalping screens. Two —5x8-ft screens separate the ore into ±2-in. products. The fines are conveyed directly to the surge bin and the oversize material passes into a hopper over each of two 7-ft cone crushers. The crushers reduce the ore to about 1½-in. product.

The head pulley and primary screens are completely housed, see Fig. 6, air being exhausted from

Fig. 8—Secondary screen enclosures, with part of surge bin and feed chutes shown, east side.

each screen housing. No air is exhausted from the enclosure over the head pulley because the ejector effect of the falling ore creates sufficient indraft around the pulley to abate escaping dust. The hopper above each 7-ft cone crusher which receives the screen oversize as well as the area surrounding the cone is housed, but no exhaust is provided. Secondary exhaust from above and below eliminates dust dissemination at these locations.

Table II. Secondary Crushing Plant Exhaust Systems

Hood Location	Hoods, No.	Volume in CFM Each	Volume in CFM Total
Symons Crushers Area, Exhaust System No. 4			
Primary screen enclosure	2	4,000	8,000
Tail of conveyor enclosures	2	2,000	4,000
Head of conveyor enclosures	2	5,000	10,000
Total	6		22,000
Screening Tower, Exhaust System No. 5			
Surge bins	2	2,250	4,500
Tail oversize conveyor enclosure	2	1,500	3,000
Head oversize conveyor enclosure	2	2,000	4,000
Tail undersize conveyor enclosure	1	2,000	2,000
Head undersize conveyor enclosure	1	3,000	3,000
Between undersize screen chutes	8	750	6,000
Total	16		22,500
Rolls and Gyratory, Exhaust System No. 6			
Rolls feed hoppers	2	1,400	2,800
Head rolls discharge	2	5,000	10,000
Tail rolls discharge	2	1,500	3,000
Head gyratory discharge	2	3,000	6,000
Tail gyratory discharge	2	1,500	3,000
Total	10		24,800
Underground Crushing, Exhaust System No. 7			
Receiving bins (present)	1		20,500
Receiving bins (proposed)	1		16,000
Gyratory crusher enclosures (proposed)	2	2,500	5,000
Vibrating grizzly feed enclosures (proposed)	2	2,500	5,000
Vibrating grizzly undersize chutes (proposed)	2	2,000	4,000
Total (proposed)	7		30,000

The major portion of the air is taken from tight-fitting housings over the conveyors receiving the screened and crushed product. Air is exhausted from the head and tail of each conveyor housing with exhaust hoods several feet from and well above feed points on the conveyors.

A total of 22,000 cfm is exhausted from the screen and conveyor enclosures by a type W Rotoclone, with volumes exhausted at each respective hood as listed in Table II.

The total measured volume slightly exceeds that of calculated design, but all branch volumes fall well within the 10 pct deviation, which is the goal set by Kennecott's ventilation design department. If at all possible, as shown in Fig. 7, hoods are located on the housing or over a conveyor transfer point so that a vertical riser precedes a horizontal run, thus minimizing the possibility of material being projected into the branch ducting.

Secondary Screening Towers: The Symons crushers product and the undersize from the primary screens are conveyed to a 360-ton capacity surge bin, situated immediately above the secondary screening plant. The ore from the bin is choke fed to 10 screens. The undersize, screened to about 2 pct on ¾-mesh screens, is conveyed directly to the shipping bins. The oversize material, including the underground ore, after passing through two gyratory crushers, is in closed circuit between the roll crushers and the secondary screens.

As illustrated in Fig. 8, the screens, as well as the oversize discharge chutes, are completely enclosed. The screen enclosures are ventilated indirectly by secondary exhaust from both the oversize and undersize product conveyor housings. The conveyors carrying the oversize from the screens are housed for the entire length of the screening tower with exhaust points at each end. The undersize product conveyor is exhausted at the head and tail with exhaust points between each screen undersize chute as indicated in Fig. 9. Through past experience this was found necessary because of the ore fall acting as a damper to air flow, which reduced the control of dust dispersed by the piston-like action of the ore as it strikes the conveyor. The exhaust points are out of line of direct ore fall, re-

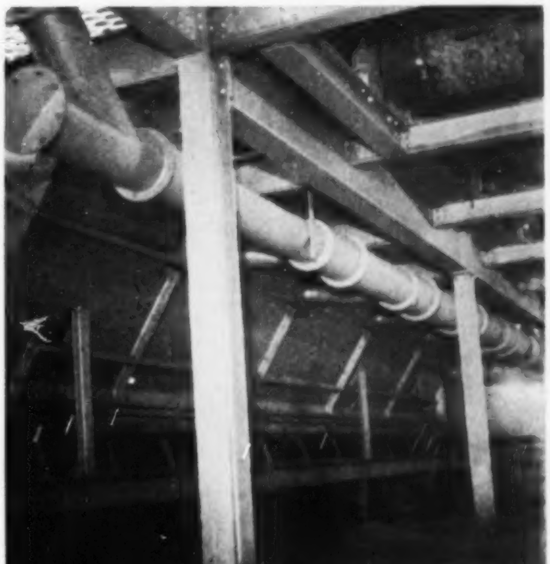

Fig. 9—Secondary screen undersize product conveyor enclosure, showing exhaust between each ore drop.

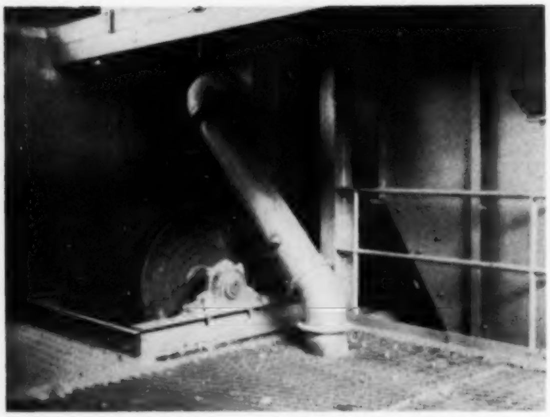

Fig. 10—Exhaust at tail of secondary screen oversize product conveyor. Note clean-up chute at end of housing and method of lighting ore chutes.

cessed, and connected to the enclosure about 5 ft above the conveyor. Fig. 10 shows an exhaust point at the tail of one of the screen oversize product conveyor enclosures. Also shown is a typical clean-up chute provided with a self-closing door. Sufficient volume is exhausted from the surge bin to maintain a negative pressure in the bin at all times. The air is exhausted from the top of the enclosed bin toward both ends. Air volumes exhausted from each hood can be found in Table II.

A volume of 22,500 cfm is exhausted by a type N Rotoclone, used as a collector-exhauster. Constant dribble from the conveyors leading to and from the screening tower made it necessary to house the take-up pulleys and provide dribble trays under the conveyors. Belt scrapers have been found to be of little value, especially when the ore is damp.

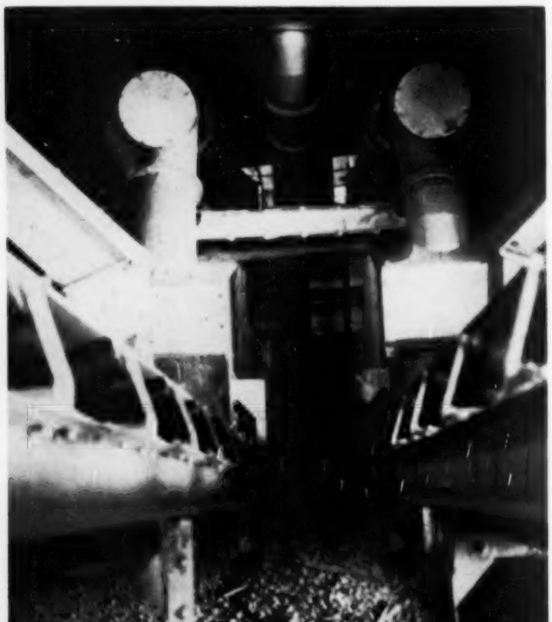

Fig. 11—Gyratory and roll crusher product feed enclosures, ducting and hoods partly shown.

Where possible the material that sluffs off the belts is discharged by chute to an exhausted enclosure.

Rolls and Gyratory Crushers: The two 72-in. roll crushers are in closed circuit with the secondary screens, handling the screen oversize which is a mixture of both pit and underground ore. Two No. 8 Gates gyratory crushers reduce the oversize material passing over the vibrating grizzlies to about a 2-in. product. Grizzly undersize material is returned to the surge bin along with that of the products from the gyratory and roll crushers.

Like the other crusher discharges, the feed points on the conveyor are completely housed, including a short section of conveyor between the gyratory and roll crushers, see Fig. 11. Openings between the enclosure and the vibrating machines are closed with old conveyor belting to reduce inordinate air inspiration.

One exhaust system using a type N Rotoclone collector-exhauster pulls 24,800 cfm from 10 hoods, 4 on each conveyor enclosure and 1 on each roll crusher feed box. The conveyor enclosure exhaust points are located at the front and rear of each ore drop.

At the present time the underground ore-receiving bins are ventilated, but no ventilation exists over the vibrating grizzlies or gyratory crushers. Designs are in the preliminary stage for additional ventilation in this area. A total of 30,000 cfm will be exhausted by existing equipment.

Housekeeping

A central vacuum cleaning system has been ordered to facilitate clean-up work in the secondary crushing plant. The unit is designed to handle two —2-in. or four 1½-in. hoses continuously. The network of piping extends to every nook and corner in the secondary as well as the conveyor way leading to the storage bins. Long radius drainage fittings with self-closing valves will be used exclusively. Material collected by the primary and secondary separators will be emptied on the final products conveyor. At least for the present, no vacuum cleaning system has been planned for the primary crushing plant. Clean-up chutes with self-closing doors have been provided at the tail of all conveyors, eliminating air losses through the doors often left open after the clean-up work is done.

Table III. Sample Mill Exhaust System — System No. 8

Hood Location	Hoods, No.	Volume in CFM
No. 1 transfer	1	3,000
No. 2 transfer, head pulley	1	1,500
No. 2 transfer, drop point	1	3,500
Sample pan-feeder	1	700
Rolls feed chute	1	1,000
Elevator discharge	1	700
Splitter enclosure	1	1,600
Total	7	12,000

Sample Mill

A sample of ore is cut periodically at the final product conveyor transfer point adjacent to the sample tower. The sample is crushed by 24-in. rolls, elevated by a bucket elevator, and a portion selected by a rotary splitter. The splitter box, rolls crusher, elevator housing, and pan-feeder are furnished with exhaust ventilation.

Included in the same system is the exhaust from two conveyor transfer points, one as the ore leaves

the secondary crushing plant and one in proximity to the sample tower. At the sample tower conveyor transfer the head pulley enclosure is exhausted as well as the ore drop, because the cutting of the sample creates an abnormal disturbance that cannot be overcome by the exhaust point lower down. A type N Rotoclone exhausts 12,000 cfm from 7 hoods as shown in Table III.

Table IV lists the collecting equipment and characteristics pertaining to each of the eight systems described.

Table IV. Collector and System Specifications for Crushing Operations Exhaust Systems

System Number	Collector	Type	Size	Total Pressure, In., W.G.	HP Required
1	Rotoclone	W	36	2.5	66.8
2	Rotoclone	W	30	2.9	44.0
3	Rotoclone	W	16	1.8	9.8
4	Rotoclone	W	30	3.3	58.2
5	Rotoclone	N	32	8.0*	40.0
6	Rotoclone	N	32	8.2*	47.0
7	Spray Chamber			Approx. 4.7*	Not Determined
8	Rotoclone	N	16	7.0*	21.6

* System loss calculated as static pressure rather than total pressure.

Since ore storage bins superstructure is open to atmosphere on all sides, there is little need for ventilation. It is only for short periods that the one man employed must enter the dusty areas, during which time he wears a dust respirator.

Plant Atmospheric Conditions

Quarterly air sampling surveys are made in all departments by the Southwest field engineer. At these times dust counts are taken near crushing and screening operations. Since the secondary crushing plant has been in operation for only a short time, complete data is lacking, but Table V gives an account of dust concentrations as sampled to date.

Table V. Dust Concentrations in Crushing Plants and Sample Mill

Sample Location	Samples, No.	Dust Concentration Million Part. Per Cu Ft High	Low	Average
Primary Crushing Plant				
Crusher operating floor	4	4.0	0.8	1.8
Crusher drive floor	7	6.4	0.8	3.3
Conveyor tunnel	5	4.8	1.6	4.0
Secondary Crushing Plant				
Symons crushers operating floor	4	4.0	1.6	2.2
Symons crushers drive floor	3	2.4	1.6	1.9
Screen tower operating floor	5	5.6	0.8	2.7
Screen tower feed floor	3	1.6	0.8	1.1
Tunnel under screens	5	8.0	1.6	3.8
Roll mill operating floor	3	5.6	0.8	2.9
Roll mill feed floor	2	8.0	1.6	4.8
Conveyor tunnel under rolls	1	7.2	7.2	7.2
Sample Mill				
Ground floor	1	1.6	1.6	1.6
Sample cutter floor	2	1.6	0.8	1.2
Junction house	1	0.8	0.8	0.8

The dust counts were taken under normal operating conditions with an M.S.A. Midget impinger, counted on Hatch cells, with a micro-projector, according to the light field technique. Even the high counts are within the 10 million particles per cu ft of air, designated as the maximum allowable concentration for silica bearing dusts, according to the Arizona code.

The proximity of the hospital and housing projects to the crushing plants was one of the deciding factors in selecting the type of dust-collecting equipment. The effluent from the stacks was to be kept to a minimum and the idea of placing collected dust back in the ore circuit as dry material was rejected. Consequently, because of its high efficiency rating, a dynamic precipitator type of collector was chosen.

Type W Rotoclones were used on the first four sytems installed. However, because water at Ray is at a premium, the last three systems in the secondary crushing plant were equipped with type N Rotoclones. Soluble copper salts, present in varying amounts in the mine ore, have created a corrosion problem in the collectors. By nature of the construction of the type N, these collectors have shown the greatest wear. It was also found necessary to apply a protective coating to all inside parts of the Rotoclones, except stainless steel parts.

If the water problem at Ray becomes more acute, some method for reclaiming the dust collector water may have to be employed. However, this is not anticipated for the near future.

A Changing Trend

Like the old sweatshop, dirty mines, mills, and crushing plants are gradually becoming a thing of the past. Management throughout the country is aware that dust, mist, and fume control contribute appreciably to better industrial relations and minimize labor turnover. Evaluating ventilation and air conditioning is somewhat more difficult than measuring metallurgical gains or losses; however, in many cases, the cost of labor turnover attributable to unsatisfactory environmental conditions will amortize ventilation systems.

It cannot be too emphatically expressed that ventilation installations are much less expensive if incorporated with the original design of the plant. Superimposing is not only more expensive as a rule, but often limits the engineer in providing adequate protection.

Notwithstanding a well-engineered transporting system, the proper selection of fans and collecting equipment may be the secret of success or failure of many types of exhaust ventilation.

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Cleaning Various Coals In A Drum-Type Dense-Medium Pilot Plant

by M. R. Geer, W. A. Olds, and H. F. Yancey

THE increase in the number of coal-cleaning plants employing dense-medium processes occurring since 1946 is especially interesting when viewed historically. Both sand and magnetite were introduced as material for heavy mediums at about the same time, sand in the Chance process in 1921 and magnetite in the Conklin process in 1922, but from that point on their records diverge. The Chance process enjoyed a steady growth from its inception, whereas no additional magnetite plants were built in the United States for over 20 years. Then, following the close of the World War II, magnetite was again introduced, this time with marked success. During the following years some 47 plants employing magnetite medium were built.

This rapid growth of dense-medium cleaning has been concurrent with widespread adoption of full-seam mining on one hand and a return to a more competitive market on the other. At a time when changing mining practice has provided cleaning plants with dirtier coal and changing market conditions have simultaneously demanded a cleaner product, the industry has through necessity turned to improved preparation. The inherently greater sharpness with which dense-medium processes can separate coal from impurity is thus helping to hold the line against ever-increasing mining costs and at the same time assisting materially in retaining badly needed markets.

Although dense-medium cleaning unquestionably offers a distinct advantage when the washing problem is difficult, other methods can provide almost equally high efficiency when the coal is easy to wash. Moreover, fine coal cannot yet be treated by heavy medium in a proved process, although the Driessen cyclone is in the pilot-plant stage.

Most of the present types of dense-medium equipment have been in use only a few years, and the dearth of information in the literature concerning their performance characteristics is entirely understandable. Nevertheless this information is necessary if the process is to be intelligently applied to individual cleaning problems. Without data on the efficiency of a process in a particular type of separation, it is difficult to assess the advantage to be expected from it. Similarly, the role of particle size in heavy-medium separation is important in some cases, yet there is little published information on this aspect. To mention only one more of the numerous points on which essential information is lacking, the bearing of medium characteristics on performance has been discussed only in qualitative terms.

It was with the hope of providing information on some of these points that the Bureau of Mines built a dense-medium pilot plant for cleaning coal at its Northwest Experiment Station in Seattle in 1950. The plant has been operated continuously since that time, and over 50 runs have been made on 7 coals exhibiting a wide range of washability characteristics. An idea of the magnitude of this work will be gained from the fact that examination of the plant products has involved some 600 float-and-sink separations and about 2500 ash determinations.

A laboratory pilot plant is especially well adapted to investigate many aspects of performance because close control over test conditions can be exercised and because a large number of tests can be made rapidly. On the other hand, factors such as consumption of medium and other cost items can be investigated satisfactorily only in a commercial plant. Actually, the two forms of investigation should be complementary, with the laboratory work pointing the way for confirming tests in commercial units.

Pilot Plant

The dense-medium pilot plant employed for this work comprises a 24x30-in. drum-type separating vessel, a 12-in. densifier, a 12-in. magnetic separator, a 26-in. x 9-ft vibrating screen, and the necessary pumps and conveyors for handling materials. Arrangement of these units corresponds with the flowsheet used in most commercial plants, except that a thickener is not provided for the feed to the magnetic separator. Coal and refuse discharge from the separating drum to the vibrator, which is divided longitudinally down the center. Medium draining through the first 3 ft of the screen is recirculated directly to the drum. Sprays on the middle 3 ft of the screen rinse medium from the products, and the last 3-ft section is for dewatering. Dilute medium from the rinsing and dewatering sections is pumped to the magnetic separator, where magnetic solids are recovered. These are pumped to the densifier, from which they return to the medium-drainage sump by gravity through a demagnetizing coil.

The drum-type separating vessel is a scale model of a commercial unit. Feed enters axially at one end of the drum just below the surface of the bath, and float material overflows through a circular opening at the other end. Particles sinking to the bottom of the bath are picked up by lifting flights bolted to the inner wall of the drum, elevated out of the bath, and sluiced to the vibrating screen. Baffles suspended in the bath prevent float material from entering the sink-lifting flights.

About 8 gpm of medium is used to sluice the feed into the drum. An additional 15 to 24 gpm, depending upon operating conditions, is added through two pipes dipping into the bath behind the baffle on the side where the sink-lifting flights enter the bath.

The bath available for separation is 2 ft long and 13 in. wide, giving an area of 2.08 sq ft. Depth of bath from the surface to the top of the sink-lifting flights, measured vertically below the axis, is 6 in.

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Table I. Identification of Coals Tested

State	County	Mine	Bed
Alaska		Buffalo	No. 2
Alaska		Buffalo	No. 5
Montana	Cascade	Surmi	Belt Creek
Washington	King	McKay Strip	Raven
Washington	Pierce	Skookum	No. 3
Washington	Whatcom	Bellingham	Bellingham
Wyoming	Carbon	Nugget	No. 1, bottom bench

when the head over the weir is 1⅛ in. Throughputs ranging from 1.5 to 5.2 tons per hr were used in the test work, but most runs were made with a feed rate of about 1.75 tons per hr. Thus the average capacity was about 0.85 tons per sq ft per hr, or from 20 to 30 pct of that attained in commercial units of the same type. The lower throughput per unit area is necessary to provide a separating time comparable to that in a commercial vessel. A particle of float travels the 2-ft length of the small drum in 13 sec, approximately equalling the time allowed for the 6-ft length of a commercial vessel of the same type. With the same treatment time, the commercial drum obtains about three times the throughput per unit area by employing a horizontal current three times as great.

In comparing drums of such dissimilar size the ratio of coal to medium is more indicative of bed crowding than throughput per unit area. The small drum treats 2.5 to 6.7 lb of feed per gal of medium, while the large size treats 4 to 6. Thus, from the standpoint of both the time available for separation and the degree of crowding in the bath where separation takes place, the laboratory unit appears to be comparable to its commercial-size counterpart.

The most serious shortcoming of the small drum is its inability to deal with the full range of sizes normally treated in a commercial-size unit. Owing largely to the shallowness of the bath, and the comparatively low head of medium over the discharge weir required to insure adequate treatment time, coal coarser than 2 in. square hole size cannot be handled. All but a few of the results presented in this report were obtained on feeds of 2 to ¼ in., square hole; in the remaining tests, the bottom size was ⅛ in., square hole.

Test Procedure

In most commercial plants, where the same coal or the same mixture of coals always is treated, the character of the medium required for satisfactory plant operation is not subject to great change. Moreover, the character of the medium used is likely to be dictated jointly by the nature of the impurities in the raw coal and the operating characteristics of the plant. In the pilot plant, however, where coals of widely different nature are cleaned, and separations ranging from 1.30 to 1.75 sp gr are required, the character of the medium is an all-important variable. Therefore the first step in preparing the pilot plant for a new operation was to select, and make provision to maintain, suitable medium.

The medium characteristic of principal operating importance is stability. If the medium is too unstable, either because the magnetite is excessively coarse or because the coal being treated supplies no clay to assist in stabilization, the differential in density between the top and bottom of the bath is undesirably high. Conversely, if the medium is too stable, either because of excessively fine magnetite or the presence of an undue amount of clay, the small density differential generally desired is not obtained. For separations at 1.55 sp gr or higher, commercial grade A magnetite, 60 to 75 pct through 325-mesh, proved satisfactory, although with some coals containing large amounts of clay a high medium-cleaning rate was needed to preserve the desired density differential in the bath. Grade B magnetite, 90 pct through 325 mesh, was required in treating coals free of clay at a sp gr of 1.40 or less. In the range between 1.40 and 1.55, either mixtures of grade A and B were used or the coarser fractions of grade A were removed from the medium by classification to obtain the desired stability. The subjects of medium stability and medium contamination are dealt with more completely elsewhere in the report.

With suitable medium circulating in the plant, coal prepared to the desired size is fed to the drum. If sufficient coal is available, up to 10 tons is run through the plant for a test, giving an operating period of about 5 hr. Often, however, the number of tests under different operating conditions desired for a coal necessarily limits the duration of the individual tests to about a half hour each. Regardless of the duration of the run, all of the washed coal and refuse is weighed to determine yield and is sampled for subsequent examination.

The washed coal and refuse samples were subjected to float-and-sink tests at each 0.1 interval of sp gr from 1.30 to 1.80. Each individual density

Table II. Specific-gravity Analyses of Coals Tested

				Cumulative	
Coal	Sp Gr	Wt, Pct	Ash, Pct	Wt, Pct	Ash, Pct
Buffalo No. 2	Under 1.30	58.9	2.2	58.9	2.2
	1.30 to 1.40	21.9	8.7	80.8	4.0
	1.40 to 1.50	6.0	22.9	86.8	5.3
	1.50 to 1.60	4.3	32.5	91.1	6.6
	1.60 to 1.70	2.2	38.8	93.3	7.3
	Over 1.70	6.7	61.9	100.0	11.0
Buffalo No. 5	Under 1.30	49.4	1.7	49.4	1.7
	1.30 to 1.40	13.6	10.9	63.0	3.7
	1.40 to 1.50	5.8	24.1	68.8	5.4
	1.50 to 1.60	4.2	33.1	73.0	7.0
	1.60 to 1.70	3.9	42.1	76.9	8.8
	1.70 to 1.80	3.0	49.1	79.9	10.3
	Over 1.80	20.1	69.6	100.0	22.2
Surmi	Under 1.30	1.2	2.6	1.2	2.6
	1.30 to 1.40	37.6	8.4	38.8	8.2
	1.40 to 1.50	23.9	18.0	62.7	11.9
	1.50 to 1.60	11.0	26.8	73.7	14.2
	1.60 to 1.70	6.2	34.6	79.9	15.8
	1.70 to 1.80	3.9	42.0	83.8	17.0
	Over 1.80	16.2	59.0	100.0	23.8
Raven	Under 1.30	6.1	3.7	6.1	3.7
	1.30 to 1.40	47.3	13.0	53.4	11.9
	1.40 to 1.50	21.0	25.4	74.4	15.7
	1.50 to 1.60	5.4	37.1	79.8	17.2
	1.60 to 1.70	6.1	47.2	85.9	19.3
	1.70 to 1.80	5.2	57.8	91.1	21.5
	Over 1.80	8.9	73.1	100.0	26.1
Skookum	Under 1.30	9.8	5.2	9.8	5.2
	1.30 to 1.40	28.0	13.9	37.8	11.6
	1.40 to 1.50	17.1	24.7	54.9	15.7
	1.50 to 1.60	6.5	34.1	61.4	17.7
	1.60 to 1.70	3.8	42.9	65.2	19.1
	1.70 to 1.80	3.0	50.1	68.2	20.5
	Over 1.80	31.8	74.4	100.0	37.6
Bellingham	Under 1.30	0.3	3.7	0.3	3.7
	1.30 to 1.40	53.6	10.9	53.9	10.9
	1.40 to 1.50	22.3	21.4	76.2	13.9
	1.50 to 1.60	8.1	33.1	84.3	15.8
	1.60 to 1.70	4.2	43.1	88.5	17.1
	1.70 to 1.80	2.6	51.5	91.1	18.1
	Over 1.80	8.9	72.4	100.0	22.9
Nugget	Under 1.30	14.3	5.2	14.3	5.2
	1.30 to 1.40	60.3	10.0	74.6	9.1
	1.40 to 1.50	8.9	24.4	83.5	10.7
	1.50 to 1.60	5.5	35.9	89.0	12.3
	1.60 to 1.70	3.6	45.9	92.6	13.6
	Over 1.70	7.4	66.1	100.0	17.5

Table III. Specific-gravity Analyses of Individual Sizes of Bellingham Washed Coal

Size, In.	Sp Gr	Wt, Pct	Ash, Pct	Cumulative Wt, Pct	Cumulative Ash, Pct
2 to 1	Under 1.30	0.0		0.0	
Wt, 23.3 pct	1.30 to 1.40	64.5	11.6	64.5	11.6
	1.40 to 1.50	32.2	21.6	96.7	14.9
	1.50 to 1.60	3.3	29.3	100.0	15.4
	1.60 to 1.70	0.0		100.0	15.4
	1.70 to 1.80	0.0		100.0	15.4
	Over 1.80	0.0		100.0	15.4
1 to ½	Under 1.30	0.2	4.7	0.2	4.7
Wt, 41.8 pct	1.30 to 1.40	72.7	11.1	72.9	11.1
	1.40 to 1.50	23.7	21.4	96.6	13.6
	1.50 to 1.60	3.4	30.8	100.0	14.2
	1.60 to 1.70	0.0		100.0	14.2
	1.70 to 1.80	0.0		100.0	14.2
	Over 1.80	0.0		100.0	14.2
½ to ¼	Under 1.30	0.7	3.3	0.7	3.3
Wt, 24.3 pct	1.30 to 1.40	58.0	10.3	58.7	10.2
	1.40 to 1.50	33.2	21.7	91.9	14.4
	1.50 to 1.60	7.7	32.1	99.6	15.7
	1.60 to 1.70	0.4	40.3	100.0	15.8
	1.70 to 1.80	0.0		100.0	15.8
	Over 1.80	0.0		100.0	15.8
¼ to 0	Under 1.30	0.6	3.3	0.6	3.3
Wt, 10.6 pct	1.30 to 1.40	70.2	9.4	70.8	9.3
	1.40 to 1.50	21.6	20.1	92.4	11.9
	1.50 to 1.60	6.3	30.9	98.7	13.1
	1.60 to 1.70	0.9	40.3	99.6	13.3
	1.70 to 1.80	0.2	45.0	99.8	13.4
	Over 1.80	0.2	50.9	100.0	13.5
Composite	Under 1.30	0.3	3.7	0.3	3.7
Wt, 100.0 pct	1.30 to 1.40	66.9	10.9	67.2	10.9
	1.40 to 1.50	27.9	21.4	95.1	14.0
	1.50 to 1.60	4.7	31.1	99.8	14.8
	1.60 to 1.70	0.2	40.3	100.0	14.8
	1.70 to 1.80	0.0		100.0	14.8
	Over 1.80	0.0		100.0	14.8

fraction obtained in this way was then screened at 1, ½, and ¼ in. to provide information on the character of the separation effected in the individual sizes of the feed. Ash analyses were made on all size-density fractions.

Coals Tested

Table I identifies coals treated in the pilot plant. Table II shows specific-gravity analyses.

The two Alaska coals are from the Buffalo mine in the Matanuska field. The coal from the No. 2 bed is the cleanest of any tested. In raw form it contains only 11.0 pct ash and 6.7 pct of impurity heavier than 1.70 sp gr. The impurity is largely shale and sandy shale, neither of which distintegrates appreciably in water or is noticeably tabular in shape. The Buffalo No. 5 coal contains over twice as much ash and 20.1 pct impurity heavier than 1.80 sp gr. A substantial part of this impurity is carbonaceous shale, the characteristic particle shape of which was tabular. Both Buffalo coals are relatively easy to wash. The cleanest portion of both coals is lighter than 1.30 sp gr, neither contains an unusually large proportion of material of intermediate density, and even in the No. 5 coal the percentage of heavy impurity is not unduly high.

The Surmi coal, from the Belt Creek bed in Cascade County, Mont., is somewhat more difficult to clean than the Buffalo No. 5. It contains about twice as much material in the range from 1.50 to 1.70 sp gr, and only 1 pct is lighter than 1.30 sp gr. The impurity heavier than 1.80 sp gr, amounting to 16.2 pct, is largely heavy bone. In fact, this coal is singularly free of shale, and contains no clay. The bone does not differ from the coal in particle shape.

The Raven coal, which originated from a strip mine in King County, Wash., is much like Surmi in that the clean coal bulks largely in the range from 1.30 to 1.50 sp gr. The 8.9 pct impurity heavier than 1.80 sp gr present is not in itself formidable. However, this heavy impurity is almost entirely shale and heavy bone, both of which have a distinctly tabular particle shape that renders them difficult to separate.

The Skookum coal, from Pierce County, Wash., is unquestionably the most difficult coal of the group to clean. The clean coal in this bed is much more friable than the shales with which it is associated and therefore concentrates in the sizes smaller than ⅛ in., which were not treated by heavy medium. The feed for the heavy-medium tests analyzed 37.6 pct ash, and contained 31.8 pct impurity heavier than 1.80 sp gr. This amount of impurity in itself constitutes a washing problem of more than usual difficulty. In addition, the coal contains more material of intermediate density than usual, and the proportion of clean coal lighter than 1.30 sp gr, 9.8 pct, is low.

The Bellingham coal, from Whatcom County, Wash., contains almost no material lighter than 1.30 sp gr. Nearly 75 pct is in the range from 1.30 to 1.50. The 8.9 pct impurity heavier than 1.80 sp gr is largely clay that disintegrates readily in water, thus adding somewhat to the difficulty of cleaning.

The Nugget coal is from the bottom bench of the No. 1 bed in a strip mine located at Hanna, Wyo. It contains only 7.4 pct impurity heavier than 1.70 sp gr, largely bone and carbonaceous shale. Sixty percent of the coal occurs in the range 1.30 to 1.40 sp gr, an additional 14 pct being lighter than 1.30.

Evaluation of Performance

Dividing all washed coal and refuse samples into seven specific-gravity fractions, subdividing each of these fractions into four size groups, and finally analyzing every size-density component for ash content is a time-consuming, costly procedure, but nothing short of this method provides the complete, basic data essential in evaluating the performance of any cleaning unit. Naturally space limitations

Table IV. Specific-gravity Analyses of Individual Sizes of Bellingham Refuse

Size, In.	Sp Gr	Wt, Pct	Ash, Pct	Cumulative Wt, Pct	Cumulative Ash, Pct
2 to 1	Under 1.30	0.0		0.0	
Wt, 18.9 pct	1.30 to 1.40	0.0		0.0	
	1.40 to 1.50	1.3	23.7	1.3	23.7
	1.50 to 1.60	39.9	34.5	41.2	34.2
	1.60 to 1.70	23.1	43.4	64.3	37.5
	1.70 to 1.80	11.5	51.2	75.8	39.6
	Over 1.80	24.2	63.8	100.0	45.4
1 to ½	Under 1.30	0.0		0.0	
Wt, 32.3 pct	1.30 to 1.40	0.0		0.0	
	1.40 to 1.50	0.7	24.6	0.7	24.6
	1.50 to 1.60	27.9	35.3	28.6	35.0
	1.60 to 1.70	23.8	43.4	52.4	38.8
	1.70 to 1.80	13.8	52.2	66.2	41.6
	Over 1.80	33.8	68.2	100.0	50.6
½ to ¼	Under 1.30	0.0		0.0	
Wt, 26.3 pct	1.30 to 1.40	0.1	10.1	0.1	10.1
	1.40 to 1.50	0.3	23.6	0.4	20.2
	1.50 to 1.60	13.6	35.3	14.0	34.9
	1.60 to 1.70	23.6	43.2	37.6	40.1
	1.70 to 1.80	17.1	51.6	54.7	43.7
	Over 1.80	45.3	70.3	100.0	55.7
¼ to 0	Under 1.30	0.0		0.0	
Wt, 22.5 pct	1.30 to 1.40	0.3	7.7	0.3	7.7
	1.40 to 1.50	0.4	21.6	0.7	15.6
	1.50 to 1.60	5.2	32.9	5.9	30.9
	1.60 to 1.70	9.6	41.8	15.5	37.6
	1.70 to 1.80	8.6	50.5	24.1	42.2
	Over 1.80	75.9	79.0	100.0	70.1
Composite	Under 1.30	0.0		0.0	
Wt, 100.0 pct	1.30 to 1.40	0.1	8.9	0.1	8.9
	1.40 to 1.50	0.6	23.7	0.7	21.5
	1.50 to 1.60	21.3	34.9	22.0	34.5
	1.60 to 1.70	20.4	43.2	42.4	38.7
	1.70 to 1.80	13.1	51.6	55.5	41.8
	Over 1.80	44.5	72.5	100.0	55.4

Table V. Specific-gravity Analyses of Individual Sizes of Bellingham Feed

Size, In.	Sp Gr	Wt, Pct	Ash, Pct	Cumulative Wt, Pct	Cumulative Ash, Pct
2 to 1	Under 1.30				
Wt, 22.4 pct	1.30 to 1.40	53.6	11.6	53.6	11.6
	1.40 to 1.50	27.0	21.6	80.6	14.9
	1.50 to 1.60	9.5	33.0	90.1	16.9
	1.60 to 1.70	3.9	43.4	94.0	18.0
	1.70 to 1.80	1.9	51.2	95.9	18.6
	Over 1.80	4.1	63.8	100.0	20.5
1 to ½	Under 1.30	0.2	4.7	0.2	4.7
Wt, 39.9 pct	1.30 to 1.40	60.9	11.1	61.1	11.1
	1.40 to 1.50	20.0	21.4	81.1	13.6
	1.50 to 1.60	7.4	33.6	88.5	15.3
	1.60 to 1.70	3.8	43.4	92.3	16.5
	1.70 to 1.80	2.2	52.2	94.5	17.3
	Over 1.80	5.5	68.2	100.0	20.1
½ to ¼	Under 1.30	0.6	3.3	0.6	3.3
Wt, 24.7 pct	1.30 to 1.40	45.7	10.3	46.3	10.2
	1.40 to 1.50	26.2	21.7	72.5	14.4
	1.50 to 1.60	9.0	33.1	81.5	16.4
	1.60 to 1.70	5.3	43.0	86.8	18.1
	1.70 to 1.80	3.6	51.6	90.4	19.4
	Over 1.80	9.6	70.3	100.0	24.3
¼ to 0	Under 1.30	0.4	3.3	0.4	3.3
Wt, 13.0 pct	1.30 to 1.40	45.9	9.4	46.3	9.3
	1.40 to 1.50	14.2	20.1	60.5	11.9
	1.50 to 1.60	5.9	31.5	66.4	13.6
	1.60 to 1.70	3.9	41.6	70.3	15.2
	1.70 to 1.80	3.2	50.2	73.5	16.7
	Over 1.80	26.5	78.8	100.0	33.2
Composite	Under 1.30	0.3	3.7	0.3	3.7
Wt, 100.0 pct	1.30 to 1.40	53.6	10.9	53.9	10.9
	1.40 to 1.50	22.3	21.4	76.2	13.9
	1.50 to 1.60	8.1	33.1	84.3	15.8
	1.60 to 1.70	4.2	43.1	88.5	17.1
	1.70 to 1.80	2.6	51.5	91.1	18.1
	Over 1.80	8.9	72.4	100.0	22.9

require that such voluminous data be presented in only summary form in a report of this kind, but an example of the complete information for a test in the pilot plant is presented in this section to illustrate the methods of interpreting and summarizing data employed in the balance of the report.

Tables III and IV, respectively, show the specific-gravity analyses of the individual-size fractions of washed coal and refuse produced in a test on Bellingham coal. Table V shows comparable data for the feed, calculated from the analyses of the two products; such calculated feeds were used exclusively in the present work to eliminate as completely as possible the influence of degradation.

The overall result of this test was to produce a washed coal of 14.8 pct ash from a feed of 22.9 pct, with a yield of 80.0 pct; the 20.0 pct refuse rejected analyzed 55.4 pct ash. Examination of the specific-gravity analyses of washed coal shown in Table III shows that the 2 to 1-in. and 1 to ½-in. sizes were entirely free of material heavier than 1.60 sp gr. The ½ to ¼-in. size contained 0.4 pct of such material, but its presence increased the ash content of that size by only 0.1 pct and had no effect on the ash content of the washed coal as a whole. Only in the degradation material finer than ¼ in. was there any impurity heavier than 1.70 sp gr.

Similarly, examination of the data for refuse in Table IV discloses that only in the coarsest size group was there more than 1 pct of coal lighter than 1.50 sp gr. Considering all sizes together, less than 0.5 pct of the refuse was coal analyzing 14.8 pct ash, the quality of the washed coal produced.

Table VI summarizes the ash reduction effected in each size of the raw coal, and shows the corresponding Fraser and Yancey efficiency values. Efficiency, as the term is used in this report, is the ratio, expressed in percentage, of the yield of washed coal actually obtained to the yield of float coal of the same ash content shown to be present in the feed by specific-gravity analysis. The efficiency for all sizes combined was 99.9 pct, while that for the ½ to ¼-in. fraction was 99.5 pct. The value of 98.6 pct for the ¼-in. to 0 material is not particularly significant because this size is formed largely through degradation, much of which takes place after the coal leaves the separating bath.

Also presented in Table VI are the distribution data for the test, which shows what percentage of each density fraction of the feed was recovered in the washed coal. These data are calculated from the specific-gravity analyses of the washed products. In the two coarsest size fractions there was perfect recovery of all coal lighter than 1.40 sp gr, and perfect elimination of all material heavier than 1.60. In the ½ to ¼-in. fraction recovery of coal was almost perfect, but elimination of impurity was impaired slightly; 6.1 pct of the 1.60 to 1.70 sp gr fraction entered the washed coal. The influence of particle size in the heavy-medium bath is shown most clearly in the treatment of material of intermediate density. In the 1.50 to 1.60 sp-gr fraction, for example, the percentage recovery increased progressively with decrease in particle size, from 29.1 in the 2 to 1-in. group to 67.6 in the material of ½ to ¼-in.

Fig. 2 shows the distribution curve obtained by plotting the distribution data for the combined sizes of feed. The method of plotting this curve and its significance has been discussed previously.[1] Throughout the present report the specific gravity of separation is that at which the distribution curve crosses the 50 pct ordinate, indicating that material of this specific gravity was divided equally between washed coal and refuse. As shown by the curve, the specific gravity of separation for all sizes combined was 1.55. The two coarsest fractions of the feed were separated at 1.54, while the ½ to ¼-in. material was separated at 1.57.

The distribution curve also provides a convenient means for characterizing the sharpness with which

Table VI. Summary of Performance of Pilot Plant on Bellingham Coal

Item	Size Fraction, In. 2 to 1	1 to ½	½ to ¼	¼ to 0	Composite 2 to 0
Screen analysis of feed, pct	22.4	39.9	24.7	13.0	100.0
Screen analysis of washed coal, pct	23.3	41.8	24.3	10.6	100.0
Screen analysis of refuse, pct	18.9	32.3	26.3	22.5	100.0
Ash content of feed, pct	20.5	20.1	24.3	33.2	22.9
Ash content of washed coal, pct	15.4	14.2	15.8	13.5	14.8
Ash content of refuse, pct	45.4	50.6	55.7	70.1	55.4
Yield of washed coal, pct	83.2	83.8	78.7	65.2	80.0
Theoretical yield, pct*	83.3	83.8	79.1	66.1	80.1
Recovery efficiency, pct	99.9	100.0	99.5	98.6	99.9
Distribution data, pct recovered in washed coal					
Under 1.30	100.0	100.0	100.0	100.0	100.0
1.30 to 1.40	100.0	100.0	99.9	99.6	100.0
1.40 to 1.50	99.2	99.4	99.7	98.9	99.5
1.50 to 1.60	29.1	38.8	67.6	69.7	46.9
1.60 to 1.70	0.0	0.0	6.1	15.7	3.8
1.70 to 1.80	0.0	0.0	0.0	4.9	0.0
Over 1.80	0.0	0.0	0.0	0.6	0.0
Specific gravity of separation	1.54	1.54	1.57	1.59	1.55
Error area	6	9	18		15
Sink in washed coal, feed, pct†	1.2	1.2	1.6	1.1	1.2
Float in refuse, feed, pct†	2.2	1.7	1.8	1.5	2.4
Total misplaced material†	3.4	2.9	3.4	2.6	3.6

* Float-and-sink yield at ash content of washed coal.
† At specific gravity of separation.

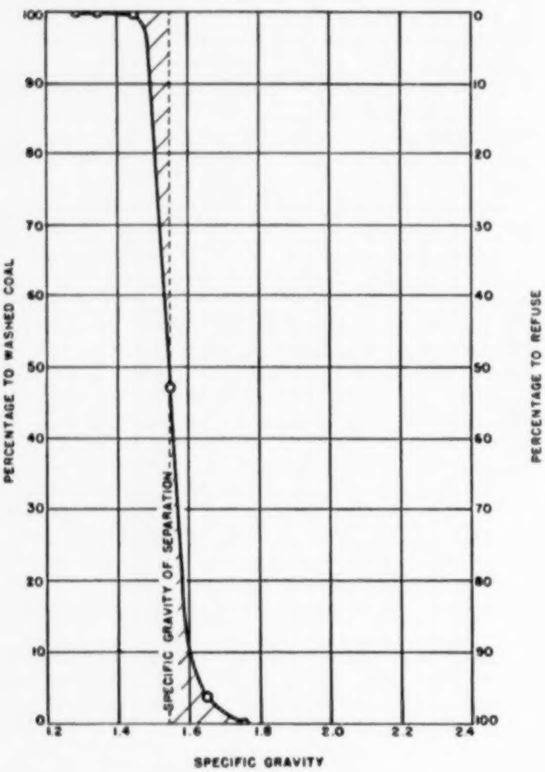

Fig. 1—Distribution curve for test on 2 to ¼-in. Bellingham coal.

the separation between coal and impurity was made. The curve representing a perfect separation would be composed of three straight lines; at densities less than the specific gravity of separation this perfect curve would show 100 pct recovery in the washed product, and at higher densities it would indicate 100 pct rejection in the refuse product. The area between the distribution curve actually obtained and the curve representing a perfect separation is thus a measure of the sharpness of separation. This area, when the curve is plotted to a fixed scale, is termed *error area*. In Fig. 2 it is indicated by cross hatching. The error area for all sizes combined in this test was 15. By particle size, it varied from 6 in the 2 to 1-in. fraction to 18 in the material of ½ to ¼-in.

In a general way, error area is a characteristic of a particular cleaning device and is not influenced directly by the specific-gravity composition of the coal being cleaned. For a given device it varies through only a relatively narrow range. Recovery efficiency, on the other hand, is determined jointly by the characteristics of the cleaning device, the density at which the separation is made, and the specific-gravity composition of the raw coal. Thus it varies over a comparatively wider range, depending on the coals treated.

The percentages of misplaced material given in Table VI, sink in the washed coal and float in the refuse at the specific gravity of separation, are expressed in terms of feed. Like recovery efficiency, misplaced material varies with the character of the raw coal and the density of the separation.

Summary of Performance Data

Of the more than 50 test runs made in the pilot plant thus far, the results of 10 were selected for presentation in Table VII to illustrate the full range of performance obtained. Six of these tests were made at specific gravities of 1.55 or higher, and four were made at 1.41 to 1.31; the results are thus divided into high and low-density groups.

In the high-density group, the results for the Nugget and Raven coals are clearly inferior, particularly as judged by error area and distribution data. The poor results obtained with these two coals are thought to be due to degradation, and will be discussed in detail under that heading. Considering the other four coals, however, the recovery efficiencies varied from 98.6 to 99.9 pct, and the corresponding spread in misplaced material was from 0.9 to 3.6 pct. Error areas ranged from 15 to 21. There was virtually complete recovery of material lighter than 1.50 sp gr in all coals, but the elimination of heavy impurity was less perfect. The percentage of sink 1.80 impurity entering the washed product ranged from nothing with Bellingham to 0.6 pct with Surmi. This imperfect rejection of impurity deserves explanation, especially as to why it varies with different coals.

With the Wilkeson No. 3 coal, the 0.2 pct of sink 1.80 material reporting to the washed product was all finer than ⅛ in.; impurity coarser than that size was eliminated completely. Similarly, the impurity entering the washed coal in the Surmi and Buffalo tests was largely finer than ¼ in. The smaller sizes of impurity are particularly difficult to remove completely from the Surmi coal because they are composed almost entirely of bone rather than heavy shale. The impurity in the Buffalo coal is predominately flaky, carbonaceous shale that is harder to remove both because of particle shape and relatively low density. Thus, although particle size and shape are of less importance in dense-medium cleaning than in other methods, they still are factors of some influence.

The group of low-density tests recorded in Table VII were made at densities well below those employed in ordinary washery operation. They are thus of interest primarily in showing the performance to be expected in the clean-coal middling split of a three-product separation, or when skimming off a superclean coal at low density. The former of these applications is widely practiced in Europe today, and probably will become increasingly important in America. The latter may be required ultimately to provide special-purpose coal for chemical utilization processes such as hydrogenation.

The Buffalo No. 2 test, made at 1.30 sp gr, is a good example of using heavy-medium treatment to skim off the cleanest fraction of a coal. Washability data indicated that a fraction analyzing 2.5 pct ash was available at a theoretical yield of 61.0 pct. Operation of the plant provided a product of that quality at a yield of 57.3 pct. The recovery efficiency was thus 93.9 pct. Perfect elimination of all material heavier than 1.50 sp gr was obtained, but the recovery of coal lighter than 1.30 was only 89.4 pct. Such a low recovery of the lightest fraction was to be expected when operating at 1.30 sp gr, however, and as judged by the error area of 13, this test provided one of the sharpest separations accomplished in the pilot plant.

The test on Wilkeson No. 3 coal at 1.36 sp gr is of special interest because of the unusually low yield, 26.2 pct, of clean coal. Normally such low yields are encountered only in re-treating refuse material. It is noteworthy that although the elimination of mate-

rial heavier than 1.60 sp gr was perfect, and the recovery of coal lighter than 1.30 was 98.5 pct, the recovery efficiency was only 89.4 pct. An efficiency of this magnitude obtained in the treatment of an ordinary coal would represent extremely poor operation, but in the treatment of a material providing such a low yield of clean coal, it probably is as high as should be expected.

Lacking a direct correlation between the performance of the pilot plant and a commercial unit of the same type, and there has been no opportunity to make such a comparison, the probability of duplicating pilot-plant results in a full-size unit remains conjectural. Both the shallowness and shortness of the separating bath in the small drum would appear to militate against equaling the performance possible in a large bath. However, these limitations might be outweighed by other factors in operating a commercial plant, especially bed congestion or lack of good density control.

Indirect evidence that comparable performance is obtainable commercially is afforded by the results published by Hanson, Heinlein, and Vonfeld.[2] Whereas their drum separators are of a different type and bed congestion and retention time probably differ materially, their washing results are similar to those obtained in the pilot plant. Their distribution data for treatment of a 3 to ⅜-in. feed at 1.52 sp gr showed perfect recovery of all material lighter than 1.40, with 0.8 pct of the sink 1.80 impurity entering the washed coal. These figures correspond closely with average pilot-plant performance. Their data for a 6 to 3-in. feed showed perfect recovery of all coal lighter than 1.40 and perfect rejection of all impurity heavier than 1.60 sp gr. A separation of this sharpness was obtained in the pilot plant only with Bellingham coal. Naturally, a 6 to 3-in. feed should separate more sharply than one of 2 to ¼-in.

Influence of Degradation on Performance Criteria

As pointed out in the discussion of Table VII, the distribution data for Raven and Nugget are clearly inferior to those obtained on the other coals, particularly from the standpoint of impurity rejection. As neither mechanical defects nor faulty operating conditions existed to account for such poor performance, an explanation was sought in the character of the coals themselves. The refuse material in both coals was of normal density and ordinary particle shape; thus neither of these factors appeared to be responsible. The most reasonable answer seemed to be the influence of degradation of the washed products subsequent to the gravity separation.

In terms of rank both coals are on the line between high-volatile C bituminous and sub-bituminous A. When fresh they are more resistant to breakage than most bituminous coals, but on drying they become much more friable. If, in the handling incident to screening, sampling, and float-and-sink examination, substantial amounts of impurity were liberated from intergrown particles in the washed coal, the rejection of impurity by the plant would be made to appear poor. This hypothesis was not verified, however, until it was observed that stored Bellingham coal, high-volatile C bituminous, gave results in the plant that were distinctly poorer than those obtained on fresh coal. Here again the elimination of impurity appeared to be poor, and degradation was again blamed.

A testing procedure was devised to evaluate the influence of degradation on the stored Bellingham coal. About 1200 lb of the 2 to ¼-in. coal was separated at 1.60 sp gr on a float-and-sink bath, using the same care exercised in the normal testing of samples. The float-and-sink products from this operation were then treated exactly the same as the products discharged from the drum in the pilot plant. That is, they were passed over the vibrating screen, sampled, and then subjected to the usual float-and-sink procedure used in evaluating plant performance. In effect, a float-and-sink bath was simply substituted for the dense-medium bath of the drum separator.

The data obained in this way were then used to calculate the distribution figures shown in Table VIII. The effect of degradation was great enough to make the perfect separation of the float-and-sink bath appear poor. Even in the 2 to 1-in. size, 1.5 pct of the 1.50 to 1.60 sp gr fraction was found in the sink product, and 11.8 pct of the 1.60 to 1.70 sp gr

Table VII. Summary of Pilot Plant Performance

	High-Density Tests						Low-Density Tests			
Coal	Buffalo No. 5	Surmi	Raven	Wilkeson No. 3	Bellingham	Nugget	Buffalo No. 2	Wilkeson No. 3	Nugget	Bellingham
Test number	30	13	37	53*	41	35	31	54*	32	25
Specific gravity of separation	1.69	1.61	1.59	1.58	1.55	1.62	1.30	1.36	1.39	1.41
Pct ±0.10 material in feed†	8.2	19.3	12.3	17.3	19.8	8.4	83.1	64.4	85.6	64.8
Ash in feed, pct	22.2	22.1	26.1	37.6	22.9	15.6	11.0	38.2	17.5	16.4
Ash in washed coal, pct	8.7	13.9	17.3	17.3	14.8	12.9	2.5	9.7	8.6	10.8
Ash in refuse, pct	66.3	49.6	61.5	68.4	55.4	56.2	22.3	48.4	37.5	26.1
Yield of washed coal, pct	76.5	77.1	80.3	60.2	80.0	93.5	57.3	26.2	69.3	63.6
Theoretical yield, pct**	76.6	78.2	80.3	60.3	80.1	94.4	61.0	29.3	71.2	65.5
Recovery efficiency, pct††	99.9	98.6	100.0	99.8	99.9	99.0	93.9	89.4	97.3	97.1
Sink in washed coal, pct§	0.3	0.5	2.2	0.8	1.2	2.2	2.0	2.3	0.8	2.6
Float in refuse, pct§	0.6	0.9	1.4	1.2	2.4	0.9	8.5	3.2	4.5	7.3
Total misplaced material, pct§	0.9	1.4	3.6	2.0	3.6	3.1	10.5	5.5	5.3	9.9
Error area	19	19	38	21	15	102	13	18	14	13
Specific-gravity distribution, pct to washed coal										
Under 1.30	100.0	100.0	100.0	100.0	100.0	100.0	89.4	98.5	99.4	99.3
1.30 to 1.40	99.9	99.7	100.0	99.9	100.0	100.0	14.6	60.0	90.6	87.0
1.40 to 1.50	99.8	99.2	99.7	99.8	99.5	99.8	0.7	2.5	4.3	15.2
1.50 to 1.60	99.8	95.1	69.7	75.0	46.9	87.4	0.0	0.4	0.4	0.4
1.60 to 1.70	81.5	9.8	27.2	9.4	3.8	44.6	0.0	0.0	0.0	0.0
1.70 to 1.80	8.6	2.0	6.9	2.0	0.0	33.5	0.0	0.0	0.1	0.0
Over 1.80	0.4	0.6	1.2	0.2	0.0	18.2	0.0	0.0	0.0	0.0

* 2 to ⅜-in. feed.
† Pct of material within ±0.10 of specific gravity of separation, adjusted for 2.0 material.
** Float-and-sink yield of feed at ash content of washed coal.
†† Yield of washed coal/theoretical yield x 100.
§ At specific gravity of separation.

fraction appeared in the float product. With decreasing particle size the influence of degradation on the migration of material from one density fraction to another was even more pronounced. Considering all size together, the distribution data for the float-and-sink bath are actually inferior to the results obtained on this coal in the pilot plant when it was fresh. The error area for the float-and-sink separation was 24, while an area of only 15 was obtained on the fresh coal in the plant.

These results provide ample evidence to substantiate the contention that the poor performance of the pilot plant on Raven and Nugget coals was due to degradation. However, they have much broader significance. Many bituminous coals are more friable than the stored Bellingham material, and many of them contain at least as high a proportion of intergrown coal and impurity particles from which liberation is obtained on breakage. Thus comparable effects from degradation must be obtained in many washing tests. In fact, a similar trial showed that the degradation effect on Wilkeson No. 3, a friable coking coal containing only a normal amount of intergrown material, was an error area of 10. Although distribution data and error area are extremely useful criteria, it must be borne in mind that they are susceptible to abnormalities due to degradation. Small differences in error area are not significant unless all factors having a bearing on sharpness of separation are strictly comparable.

Table. VIII. Distribution Data for Test on Bellingham Coal in Float-and-Sink Bath

Sp Gr	Sizes Recovered in Float Coal, Pct				
	2 to 1	1 to ½	½ to ¼	¼ to 2	2 to 0
Under 1.30	100.0	100.0	100.0	100.0	100.0
1.30 to 1.40	100.0	100.0	100.0	99.8	99.9
1.40 to 1.50	100.0	99.8	99.7	99.2	99.7
1.50 to 1.60	98.5	96.6	95.5	92.6	95.8
1.60 to 1.70	11.8	16.2	16.7	18.7	16.6
1.70 to 1.80	0.0	2.8	2.6	7.5	5.6
Over 1.80	0.0	2.3	1.6	1.1	0.9
Error area	14	25	23		24

It is noteworthy that recovery efficiency is less subject to errors owing to degradation than are either error area or percentage of misplaced material. The latter two criteria are affected directly by the liberation of particles accompanying degradation. On the other hand, liberation of impurity in a washed coal does not change its ash content, and therefore affects recovery efficiency only indirectly through the resulting minor change in the yield-ash curve of the composite feed to the washing unit.

Operating Variables

A number of factors influencing sharpness of separation effected between coal and impurity in the dense-medium bath are beyond the control of the operator, because they are fixed by either the design of the separating vessel or the nature of the cleaning problem. Other factors, however, are subject to some degree of control, and hence can be regulated to provide optimum conditions in the bath. Among the more important factors in the latter category that were investigated in the course of the present work are density control, density differential between the top and bottom of the bath, amount of agitation in the bath, congestion or crowding of the coal in the bath, and medium contamination.

Controlling the density of the medium proved to be the simplest of the problems encountered. During the first year of plant operation, density was measured in the conventional manner by periodical weighing of a flask of known volume. Then a device for continuously indicating density was developed to replace the time-consuming, cumbersome procedure of manual weighing. The densimeter,[3] as it is known, is merely a 40-ft coil of ordinary garden hose suspended from one arm of a balance. Since the volume of medium in the hose is constant, any change in density produces a corresponding change in weight. Thus, if a tare weight on the balance is shifted to bring the indicating arm to 0 on the scale when the circulating medium is at the desired density, any subsequent change in density is indicated directly on the scale. The operator has merely to glance at it to know the direction and extent the density of the medium has varied from the desired value. With the use of this simple densimeter, the fluctuation in density during a test period has been reduced to thousandths rather than hundredths, and separation errors due to varying density have thus been eliminated.

Table IX. Influence of Density Differential in Bath on Performance

Item	Test No.	
	15	16
Specific gravity of separation	1.58	1.57
Difference in specific gravity between top and bottom of bath	0.00	0.05
Distribution data, pct to washed coal		
Under 1.30	100.0	100.0
1.30 to 1.40	99.0	99.8
1.40 to 1.50	96.7	99.4
1.50 to 1.60	71.7	73.0
1.60 to 1.70	21.1	2.9
1.70 to 1.80	6.0	0.3
Over 1.80	1.0	0.4
Error area	37	17

Density Differential: Thickening of the medium, with a resultant differential in density between the top and bottom of the bath, may occur, depending on the amount of agitation provided by the rotation of the drum. Differential also is influenced by the density of the medium, the fineness of the magnetite, and the amount of clay present. While the density of the medium is fixed by the requirements of the separation, all other factors are subject to regulation. Thus density differential is one of the controllable variables. When operating at specific gravities of 1.55 and higher it appears to be of considerable importance, but at specific gravities of 1.40 and lower it has no influence on performance.

The density differential is determined by comparing the specific gravity of the medium overflowing the discharge weir with that of a sample of the medium collected just above the sink-lifting flights at the discharge end of the drum. The measurement is thus an arbitrary value referring solely to the conditions at the discharge end. It is entirely possible that the differential would be higher if it were measured farther toward the feed end, but no attempt has been made to measure it along the full length of the drum. Present measurement has proved a satisfactory criterion of bath conditions.

Table IX shows the results of two companion tests made on 2 to ¼-in. Surmi coal with and without density differential in the bath. Both tests were made under identical operating conditions except

for the cleanness of the medium. In test 15 the medium contained a great deal of clay originating from a coal run previously, and as a result there was no measurable differential. For test 16 the same medium was freed of clay by routing it through the medium-cleaning circuit to develop a differential of 0.05. Under this condition the performance was clearly superior. Rejection of impurity was improved greatly, and the recovery of coal was slightly better. The error area was decreased from 37 to 17.

Table X summarizes, in terms of error area, all the test data available for which operating conditions other than density differential were strictly comparable. These error areas are necessarily influenced to a limited extent by the nature of the impurities in the individual coals involved, because flaky or lower-density impurities are not so completely rejected as more amenable varieties under any bath condition. Nevertheless, in the high-density group of tests those of lowest error area were obtained with differentials of 0.04 or 0.05, whereas for the higher error areas no differential existed. It is equally significant that in the low-density tests differential is unimportant.

Presumably density differential allows the bed of float coal at the surface of the bath to dilate vertically. This dilation or distention minimizes mechanical entrapment of sink particles, and thus improves the sharpness of the separation. When operating at low densities, the relative buoyancy of the coal is less. Therefore there is less tendency to form a compact bed favoring mechanical entrapment; hence density differential is less important. An analogous condition can be observed in float-and-sink testing. When a sample is immersed in a high-density bath the buoyancy of the coal is so great that a firm mat or layer of float forms at the surface immediately; vigorous stirring is required to release entrapped particles of sink material. With a low-density bath, however, the particles are less buoyant, and the layer of float is distended by only gentle agitation.

Table X. Comparison of Tests with High and Low Density Differentials

Test No.	Separation, Sp Gr	Sp Gr* Differential	Error Area
High-density tests			
41	1.55	0.05	15
40	1.57	0.04	16
16	1.57	0.05	17
53	1.58	0.05	21
14	1.60	0.00	28
39	1.50	0.00	35
15	1.58	0.00	37
37	1.59	0.00	38
Low-density tests			
34	1.35	0.00	11
25	1.41	0.03	13
27	1.39	0.03	14
29	1.39	0.00	14
32	1.39	0.00	14
33	1.39	0.00	14
50	1.39	0.05	15
51	1.39	0.04	17

* Measured between top of bath and just above sink-lifting flights at discharge of drum.

Drum Speed: The primary function of drum rotation is to elevate sink material, but an important secondary effect is the agitation of the bath. Thus the speed of drum rotation can be used to control agitation, which in turn assists in controlling the density differential in the bath. An undesirably high differential occasioned by over-cleaning the medium can be reduced by increasing the drum speed, while with a more stable medium, differential can be increased by slowing down the drum.

Obviously, violent agitation of the bath would militate against sharpness of separation because of the uncontrolled, stray currents produced. Conversely, however, too quiescent a bath promotes mechanical entrapment; returning to the analogy of the float-and-sink test, some stirring is essential. Between these two extremes there is considerable latitude in the amount of agitation that can be tolerated without impairing the separation.

Table XI. Relation Between Drum Speed and Sharpness of Separation

	Test No. 13	16	11
Drum speed, rpm	1	2	5
Specific gravity of separation	1.61	1.57	1.55
Distribution data, pct to washed coal			
Under 1.30	100.0	100.0	100.0
1.30 to 1.40	99.7	99.8	99.7
1.40 to 1.50	99.2	99.4	95.1
1.50 to 1.60	95.1	73.0	45.9
1.60 to 1.70	9.8	3.3	9.2
1.70 to 1.80	2.0	0.5	2.0
Over 1.80	0.6	0.3	0.4
Error area	19	17	22

Table XI shows the distribution data and error areas for three tests made on the same coal at drum speeds of 1, 2, and 5 rpm. Size of feed, feed rate, and density differential were the same in all three tests, and the range in specific gravity of separation from 1.55 to 1.61 is not great enough to preclude direct comparison. From the standpoint of both distribution data and error areas, the test at 2 rpm is slightly superior to the others, but the differences are not great.

At 5 rpm the surface of the bath in the drum is agitated so vigorously that small transverse waves are formed. It is somewhat surprising that such a degree of agitation does not impair the sharpness of separation. However, four tests have been made at 5 rpm and six have been made at 3 rpm, and except where the resulting density differential was insufficient, the results of these runs were not inferior to those obtained at the normal operating speed of 2 rpm. Thus the conclusion that considerable agitation can be tolerated appears to be well founded.

Owing to the higher peripheral speeds involved, a commercial-size drum might have to operate at a much lower rpm than the small laboratory unit to develop the same degree of agitation. Nevertheless, the same relation between agitation and sharpness of separation should apply.

Ratio of Medium to Coal: The bath of medium transports the float material forward through the drum and discharges it over the weir. Thus the depth of medium overflowing the weir must be great enough to remove coal at the rate required to preserve a degree of bed congestion conducive to good gravity separation. Primarily, the depth at the weir is determined by the feed rate, the percentage of float, and the size of the coal. Higher feed rate, a greater proportion of float, or coarser coal requires greater depth. Excessive depth over the weir is to be avoided, because it increases the horizontal velocity of the medium, reducing retention time.

Table XII shows clearly the results of operating the drum with an inadequate depth overflowing the weir. Tests 17 and 18 were made on identical feeds at 5.2 tons per hr, nearly three times the normal feed rate. In test 17 the usual weir depth of 1⅛-in. was preserved. This caused excessive crowding at the weir, and resulted in clean coal being forced down into the sink-lifting flights where it was lost. Increasing the flow of medium over the weir from 23 to 26 gpm in test 18 ended bed congestion, eliminated loss of clean coal, and reduced the error area of the separation from 39 to 20.

Table XII. Influence of Congestion on Sharpness of Separation

Item	Test No. 17	Test No. 18
Feed per hr, tons	5.2	5.2
Depth of medium overflowing weir, in.	1.13	1.25
Medium overflowing weir, gpm	23	26
Feed per hr per sq ft of bath, tons	2.50	2.50
Feed per gal of medium, lb	7.54	6.67
Distribution data, pct to washed coal		
Under 1.30	93.3	100.0
1.30 to 1.40	89.5	99.7
1.40 to 1.50	82.7	97.9
1.50 to 1.60	53.7	68.7
1.60 to 1.70	5.9	3.6
1.70 to 1.80	1.3	0.0
Over 1.80	0.0	0.0
Specific gravity of separation	1.56	1.57
Error area	39	20

It is noteworthy that the cleaning results in test 18 are as good as those obtained on the same coal at the normal feed rate of 1¾ tons per hr. The throughput at which performance of the plant would suffer has not been reached.

The ratio of coal to medium overflowing the weir is a measure of the degree of bed congestion, and therefore reflects the likelihood for sink particles becoming mechanically entrapped in the float product. In test 17, with 7.54 lb of coal per gal of medium, there was poorer elimination of 1.60 to 1.80 sp-gr material than in test 18 at 6.67 lb. Thus optimum congestion for maximum throughput without impaired performance appears to be 6½ to 7 lb of coal per gal for a 2 to ¼-in. feed.

Table XIII. Comparison of Tests of 2 to ¼-In. Bellingham Coal with Clean and Contaminated Media

Item	Test No. 43	Test No. 44
Specific gravity of medium	1.60	1.60
Nonmagnetic material in medium solids, pct	47.2	9.9
Specific-gravity differential in bath	0.00	0.01
Distribution data, pct to washed coal		
Under 130	100.0	100.0
1.30 to 1.40	99.9	99.9
1.40 to 1.50	99.9	99.8
1.50 to 1.60	93.4	86.6
1.60 to 1.70	29.7	23.8
1.70 to 1.80	6.0	5.6
Over 1.80	1.9	2.3
Specific gravity of separation	1.62	1.61
Error area	29	31
Recovery efficiency, pct	99.5	99.5

Contamination of Medium: The clay associated with the Bellingham coal disintegrates readily in water, and hence is a marked source of medium contamination. When the pilot plant was run on this coal, provision was made to divert sufficient medium to the medium-cleaning circuit to insure the maintenance of satisfactory quality. Cleaning only the medium from the rinsing section of the screen did not maintain the desired density differential in the bath. Moreover, it was feared that an excessive amount of clay in the medium might increase its viscosity enough to interfere with the sharpness of separation.

Once satisfactory cleaning results had been obtained with clean medium, however, a run was made with dirty medium to see how much the performance of the plant would suffer. In preparation for this run, several tons of refuse from a previous test was recirculated through the plant until most of the clay had disappeared. The medium obtained with this intentional contamination was so stable that it would not settle out on standing for hours, and magnetic separation in a Davis tube showed that 47.2 pct of the solids were nonmagnetic. Preliminary experiments had shown that when 55 pct of the medium solids were Bellingham clay a semi-gel that would scarcely pour was formed. Thus it was felt that operation of the plant with the contaminated medium would show, in exaggerated degree, the influence of viscosity.

Test 43 was made with the contaminated medium and test 44 with clean medium to provide a direct comparison. As shown in Table XIII, the distribution data and error areas for the two tests are very similar. The recovery efficiencies are identical. These tests were made on the Bellingham coal after it had been weakened by storage, and because of the degradation factor discussed previously, neither one appears to be as sharp a separation as that obtained on the fresh coal. Despite this mitigating circumstance, the performance obtained with dirty medium clearly is not significantly inferior to that with clean medium under otherwise similar operating conditions.

One test, even with badly contaminated medium, is not conclusive, and a number of writers have expressed the opinion that the viscosity of medium is a factor in performance. It would be presumptive, therefore, to conclude that viscosity is unimportant. In view of the present results, however, it would be interesting to see comparable figures for a commercial plant. Perhaps viscosity has been overrated.

At least in the pilot plant, medium contamination is more important from the standpoint of stability than from its influence on viscosity. Too stable a medium will not provide the density differential generally required in the separating bath for best cleaning results.

Summary

Seven coals exhibiting a wide range in washability characteristics have been washed in a small-scale, dense-medium pilot plant. Separations ranged from 1.30 to 1.69 sp gr. With most of the coals treated at the higher densities, virtually perfect recovery of material lighter than 1.40 sp gr was obtained. Rejection of impurity was not perfect, however, with as much as 0.6 pct of the sink 1.80 material entering the washed coal. Operation at specific gravities of 1.41 and lower was characterized by perfect elimination of impurity, but with impaired recovery of the lightest coal.

Recovery efficiencies varied from 98.6 to 99.9 pct for separations made in the usual range of washing densities. Decreased efficiencies were obtained in separations at lower densities. With one particularly dirty coal, which provided a yield of clean product amounting to only 26 pct, the efficiency was only 89.4 pct, despite the fact that the separation between coal and impurity was relatively sharp. Results

similar to this might be expected in re-treating refuse by dense medium, because of the low separating density required and the low yield of clean product.

With operation at all densities, the smaller sizes of the feed were not separated as sharply or efficiently as the coarser material. Similarly, impurities of the tabular shape were not removed as completely as others, and impurities of relatively low density such as carbonaceous shale and bone were not removed as completely as heavier varieties, such as true shale.

A differential in specific gravity of about 0.05 between the top and bottom of the separating bath gave the best cleaning results when operating at the higher densities, but was unimportant in low-density separation.

A great deal of latitude was found in the degree of agitation of the bath that could be tolerated without impairing performance. Drum speeds of 1 and 5 rpm gave cleaning results nearly as good as those obtained at 2 rpm, indicating the need for some agitation of the bath.

Comparative washing tests were made with clean and badly contaminated mediums to determine whether the viscosity of a medium influences cleaning results. Even though the contaminated medium contained 47.2 pct nonmagnetic material, largely clay, it gave washing results that were not significantly inferior to those obtained with clean medium.

Anomalous results obtained in cleaning several coals were traced to degradation of the washed products, which caused a marked increase in error area but had little influence on recovery efficiency.

Acknowledgments

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An Analysis of Mine Opening Failure By Means of Models

by Bernard York and John J. Reed

Mine opening stability was investigated by loading model openings to failure. Eight-inch plaster blocks were cast with small uniform section openings passing through the centers. After curing, the models were jacketed and loaded hydrostatically in a closed vessel. Opening size, shape, and orientation were varied to determine how these factors influence opening stability.

THE investigation described in this paper is a continuation of a project begun in 1946 by Bernard York and W. E. Bush. The experimental method used in the investigation involves the hydrostatic loading of a model mine opening to failure. Most investigators agree that this loading closely approaches actual conditions in very deep mines. The conditions are further idealized by using an essentially isotropic model material. The experiments consist of casting 8-in. plaster cubes, or 8-in. diam cylinders 8-in. long, having a relatively small uniform section hole passing through their centers. After thorough drying, the models are jacketed in a flexible plastic bag and loaded hydrostatically with water in a closed vessel. At about 300 to 500 lb per sq in. pressure the central opening fails. The fracture pattern resulting from opening failure is studied in a plane normal to the longitudinal axis of the opening. The model block is sawed through its center after loading to expose this plane and to eliminate the possibility of end effect. By changing

Fig. 1—Pressure vessel open in position to jacket the model.

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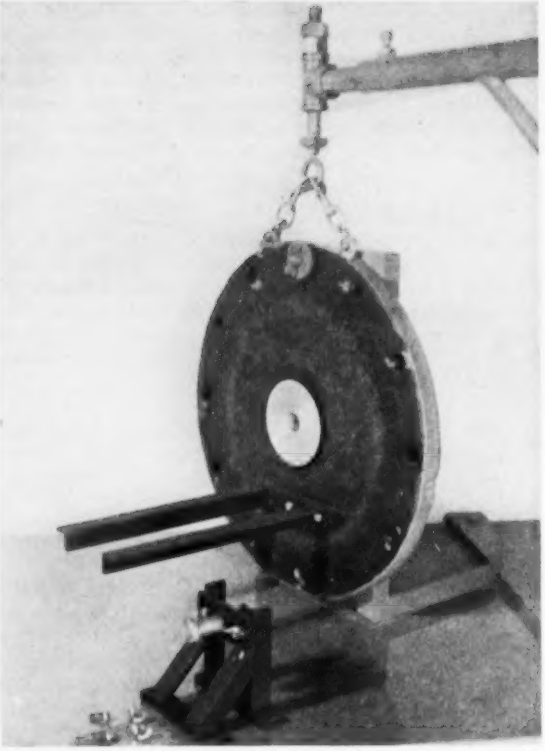

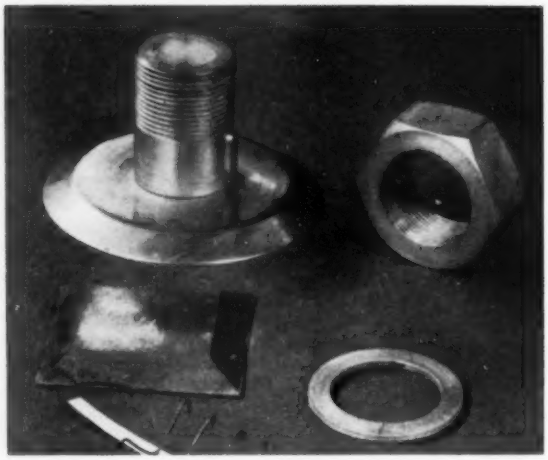

Fig. 2a (left)—Observation tube assembly in place, and Fig. 2b (right) observation tube assembly and rear cover plate.

the size, shape, and orientation of the opening it is possible to determine how the overall stability of an opening varies as a function of these factors. Once the actual manner of failure of mine openings is thoroughly understood, the means to prevent such failure may become evident. The experiments described in this paper, which are performed under the most simplified conditions, are a beginning toward this end.

In analyzing mine opening failure it is important to consider *how* an opening fails, *where* initial failure occurs, and *when* failure occurs as the load is increased. This last factor is the true measure of mine opening stability.

The results of this investigation substantiate mathematical and photoelastic analyses in demonstrating *how* mine openings will fail under hydrostatic loading. They will fail in shear. Experimental results also agree with theoretical analysis as to *where* initial failure owing to stress concentration will occur on non-cylindrical openings. However, there is a distinct difference between test results and theoretical analysis as to *when* various sizes and shapes of openings will fail in a brittle material loaded hydrostatically.

Fig. 3—Rock and unjacketed model mounted on head of pressure vessel.

As described in the section on the factors controlling opening stability, this difference can be explained in the test results on the models by consideration of probability of failure as well as the calculated maximum stresses on the openings.

If the probability of failure has such an important control on the stability of a nearly perfect cast opening in a relatively isotropic model material, then it certainly must be the major controlling factor on the stability of a prototype mine opening; because the rock is never isotropic, the opening is usually highly irregular, and, because of blasting, it is surrounded invariably by a zone of shattered material with a high failure-probability.

Using present methods and equipment, the stability of mine openings can be increased considerably by careful design of the drill rounds and blasting techniques used to create them. More trim holes of smaller diameter, loaded lightly with a low-grade bulky explosive, will result in the minimum damage to the remaining rock section.

The ideal method of driving an opening would be by boring or cutting out the rock without blasting. Shafts drilled by shot drills are an example, and have proved practical. Unfortunately, shot drills will drill only near-vertical openings, and a machine to drill or cut a horizontal opening of usable size in hard rock remains to be developed, although horizontal holes 21 in. in diam have been drilled by diamond drill. An opening driven by such means would not need to be as large as conventional mine openings because of its smooth self-supporting nature. A round opening 4 or 5 ft in diam might be quite effective. Scrapers and conveyors are capable of moving large tonnages through openings of small cross section. The development of a machine to bore or cut horizontal mine openings is worthy of the full support and interest of the mining industry.

The importance of making mine openings small, with regular cross section, without damaging the remaining rock section, has been neglected by mine operators generally. The results of this investigation demonstrate that these factors exert the major control on the stability of the resultant opening. Artificial opening support is expensive and troublesome. Therefore it is important to design the opening carefully as to size and shape for maximum stability, and then exert every effort to drive the opening in such manner as to preserve the inherent strength of the material as fully as possible.

Method of Investigation

Model Material: A model material was required for the investigation which would fail brittlely as do most rocks. A plaster mixture was found suitable, and a standard mix of 1.0 part No. 1 pottery plaster, 7.0 parts −200 mesh silica flour, and 3.1 parts water has been adopted. The average uniaxial compressive strength of this mixture is about 300 psi. By standardization and accurate control of mixing, pouring, and curing procedures very satisfactory uniformity of strength and properties is obtained from block to block.

Loading the Models: A general view of the pressure-loading equipment is shown in Fig. 1. The pressure vessel is a section of 12-in. pipe with a bolted steel cover. A rack fastened to the inside of the vessel head carries the model block, Fig. 2a. A round observation tube passes through the head of the vessel and provides a 1-in. round opening directly opposite the opening in the model, giving an unobstructed view directly inside the model opening during loading. The observation tube assembly with its bearing plate is shown detached in Fig. 2b, and in place in the vessel head, Fig. 2a. The square beveled plate shown in Fig. 2b covers the opposite end of the model opening. A spring-loaded plunger, shown detached in Fig. 2a, bears on the rear cover plate and holds the block in close contact with the plate of the observation tube until the pressure comes on. Fig. 3 shows an unjacketed model mount-

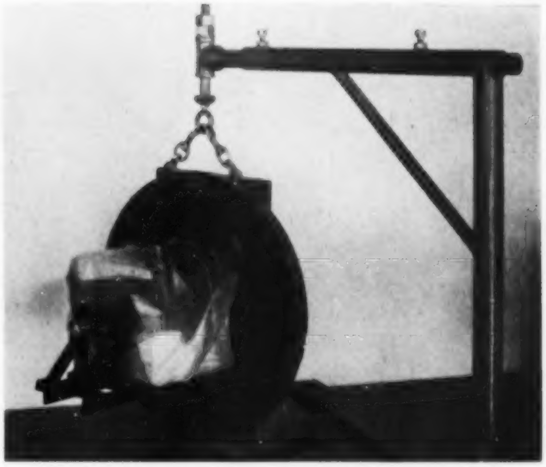

Fig. 4—Jacketed model ready to enter pressure vessel.

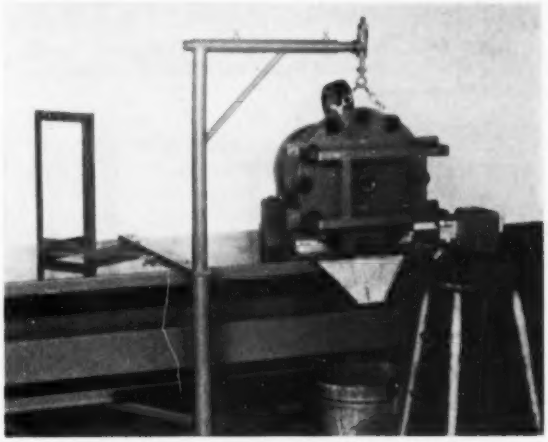

Fig. 5—Vessel closed with protective shield and light source in position.

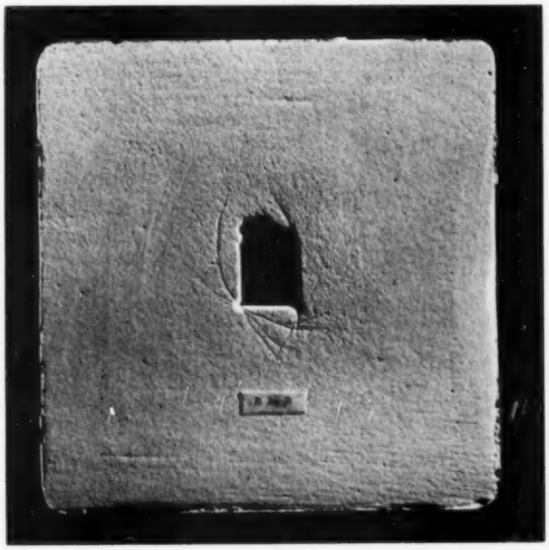

Fig. 6a—A photograph of model type R-11, which has an opening 1 in. square. Fig. 6b at left shows a close-up of the localized fracture pattern. All models are made of pottery plaster, —200 mesh silica flour, and water.

Fig. 6b—A close-up of fracture failure, model type R-11. Larger and smaller openings of similar shape failed in exactly the same manner, except that loading pressure at initial failure was lower and higher, respectively.

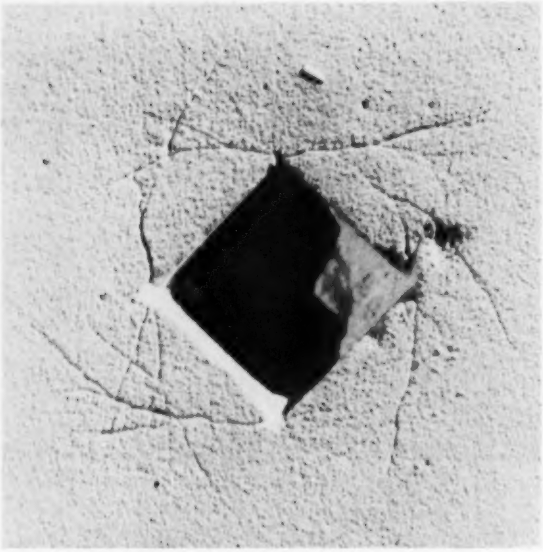

Fig. 7—Model type R-7 with opening 1 in. square. Model type R-19 has opening of similar shape, 0.35 in. square. Both models are square.

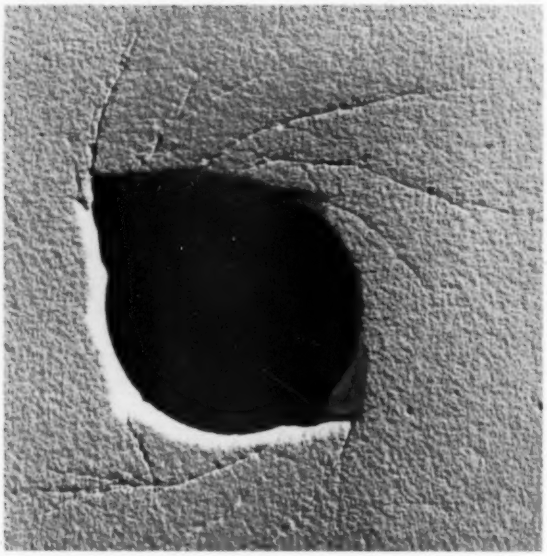

Fig. 8—A photograph of model type R-10, showing opening 1.4 in. round. The model is square. Similar models have openings 0.5 in., 0.75 in., 1 in., and 2 in. round.

ed on the rack and Fig. 4 a jacketed model ready to enter the pressure vessel. The jacket is a simple flat bag 20 by 30 in. made commercially from 0.008-in. vinyl plastic material. The bag is open at one end, and once the model is inside, the mouth of the bag is sealed for about 1 in. with vinyl-to-vinyl cement. The observation tube passes through a hole in the side of the bag, and the bag itself forms a gasket between the bearing plate and the head of the vessel. Fig. 5 shows the equipment with the vessel closed and ready to apply pressure. A rigid transparent shield protects the observer from plaster and water which might blow out through the observation tube if the jacket fails under pressure. A slide projector provides illumination within the model opening during loading.

The pump connected to the pressure vessel is operated manually, and is capable of developing pressures in excess of 2000 psi. During loading of the model, one person operates the pump and watches the pressure gage, while a second person observes the model through the observation tube. The pressure is raised very slowly at first to allow all possible air to work out of the collapsing plastic bag and escape through the observation tube. Then it is gradually increased with a short pause after each 25-lb increment. The pressure at initial failure is recorded when the first definite spalling is observed within the opening. This pressure can be determined within the nearest 25 psi, or about 7 pct of the ultimate load. Thus the limit of accuracy of the method as described is about 7 pct.

Types of Models and Openings Tested

Examples of some of the 11 types of model blocks tested during this investigation are shown in Figs. 6-10, which illustrate the fracture failures formed during testing. Two or more blocks were made of each type. A complete series of square and round openings was tested. Dimensions of the openings ranged from 0.5 to 2.0 in.

It became apparent early in the investigation that the maximum dimension of the opening was an important factor controlling the loading pressure required to cause failure. Therefore corresponding round and square openings were made, the diameter of the round openings being equal to the diagonal of the square ones. Thus the relative stability of the two shapes could be compared.

The results of testing 44 model blocks indicate that they are sufficiently uniform in strength and physical properties to compare the manner and degree of failure, and the loading pressure at failure, from block to block. Throughout the tests, duplicate blocks invariably gave duplicate results.

Only the tests of 1.4-in. openings will be de-

Fig. 9—Model type R-15, which has opening 1 in. square. The model itself is circular.

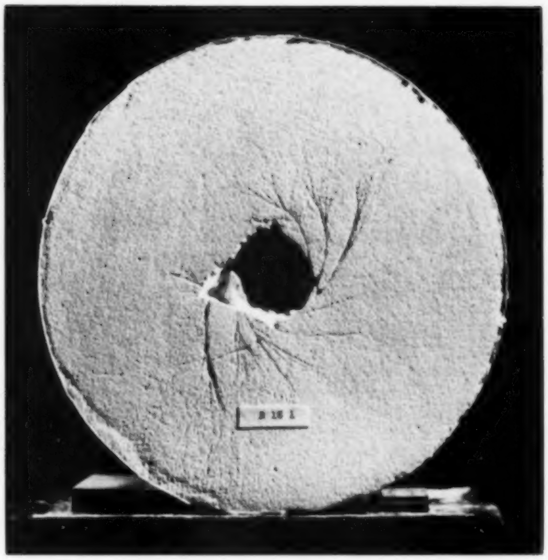

Fig. 10a—A photograph of model type R-16, which has an opening 1.4 in. round.

Fig. 10b—A close-up of the fracture failure around opening of model type R-16, shown at left.

scribed in detail. Larger and smaller openings failed in exactly the same manner except that the loading pressure at initial failure was lower and higher, respectively. The photographs of fracture failures, Figs. 6-10, were all taken on sections through the center of the blocks and normal to the axes of the openings. For two typical models, Figs. 6a and 6b and Figs. 10a and 10b, both a general picture of the entire block and a close-up of the localized fracture pattern are included.

Mathematical analysis of the stresses around an opening in a hydrostatic stress field shows that the stress tangent to the periphery of the opening is everywhere a compressive stress. This is in accordance with the theory of elasticity developed by Timoshenko. It can be shown by the theory that if the unit hydrostatic load is P, the maximum unit compressive stress at the periphery of a cylindrical opening will be $2P$. The maximum unit shear stress will be P acting on a curved plane starting at approximately 45° with the periphery of the opening. Under hydrostatic loading these stresses are the minimum on a cylindrical opening and are equal anywhere on the periphery of that opening. Around non-cylindrical openings the stresses will be lower than $2P$ in some portions and higher in others and will be highest at points where the radius of curvature is smallest.

This same problem of stress analysis can be treated experimentally by photoelastic methods, and the same result is obtained as by mathematical analysis.

The manner in which the model openings failed supports both the mathematical and experimental stress analyses with regard to *how* and *where* the openings fail. Failure occurred along planes of maximum shear stress in every test, and the fractures exhibit the spiral pattern of the curves plotting that stress. There were no signs of tension failure in any models. Fig. 11 shows fracture fragments formed by failure of 1-in. square openings. They have the characteristic shape indicating failure in shear, and because of longitudinal translation of the fracture, they are usually unbroken through the entire length of the opening.

Both the cylindrical and cubic models show the same fracture patterns on the various openings, indicating that in either model shape the opening is subject to approximately the same stress field. Initial failure in square openings occurred at corners because of stress concentration at these points.

Factors Controlling Opening Stability

Type of Loading: With existing equipment, only hydrostatic loading can be developed. With equipment capable of loading unequally on the two axes perpendicular to the axis of the central opening, it will be possible to influence opening stability by the type of loading employed.

Orientation of Opening: Under hydrostatic loading any unit plane within the model is subject to the same pressure, regardless of its orientation. Therefore any opening will be equally stable in a given hydrostatic stress field, regardless of its orientation.

Shape of Opening: According to the theory of elasticity, the shape of an opening in a hydrostatic stress field is the all-important factor that determines the maximum stress on the opening, and from

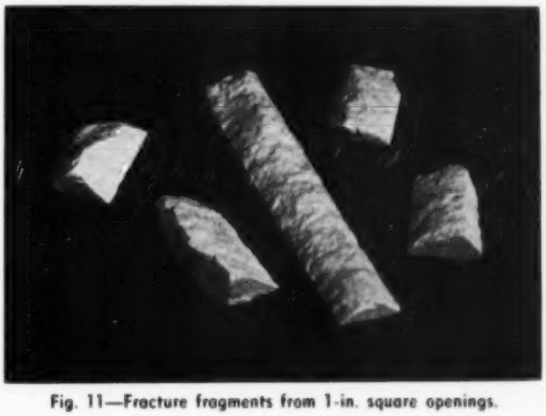

Fig. 11—Fracture fragments from 1-in. square openings.

this it is inferred generally that the shape of the opening is also the major factor controlling the stability of the opening.

The stress concentration at each corner of a true rectangle, one with sharp corners, is infinite. Even on a square opening with rounded corners, the maximum stress at the corners exceeds twice that on a circular opening.

The square openings used in this investigation all had relatively sharp corners. Therefore, according to the theory and inference cited above, there should be a wide difference between the loading at failure for a square opening and that for a round one. It appears highly significant that no such difference is apparent in the experimental results. Within the accuracy of the method, this investigation shows no appreciable difference in the loading at failure between a square opening and a round one of the same maximum dimension.

Size of Opening: The experimental results of this investigation indicate that the maximum dimension of the opening is the major factor controlling opening stability. Within the accuracy of this method, all openings of the same maximum dimension failed at the same loading pressure regardless of their shape. Fig. 12 is a graph of the test results made by plotting the greatest dimension of the opening against the loading pressure at initial failure. Results show that the loading pressure at initial opening failure will vary inversely with the greatest dimension of the opening. This result, coupled with field observations in mines, offers strong support to the argument that opening size is the major factor in controlling opening stability.

Mention should be made of the indicated compressive stress at failure for each of the various size openings tested. According to the elastic theory, this unit stress on a cylindrical opening is twice the unit loading pressure. Therefore the indicated unit compressive stress at failure on the periphery of the round openings tested varied from 634 psi on a 2-in. opening to 1034 psi on a 0.5-in. opening. This apparently anomalous result can be explained by several factors, as follows.

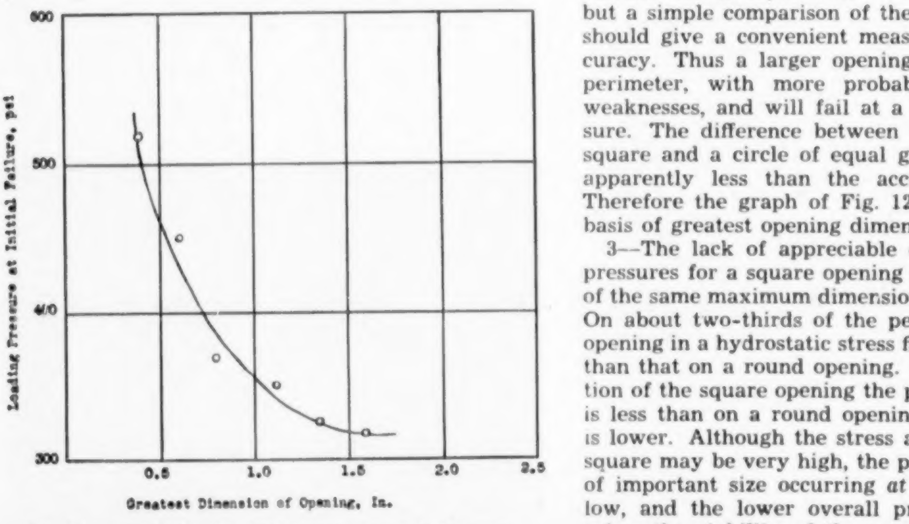

Fig. 12—A graph showing loading pressure at initial opening failure.

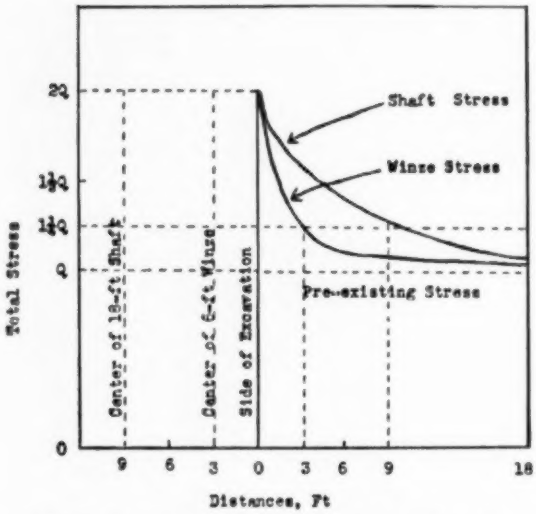

Fig. 13—Control of stress distribution by size of opening. (After Jack Spalding, *Deep Mining*, p. 91. Mining Publications, Ltd.)

1—Failure of a brittle material occurs in response to stress over a *finite length* of an incipient crack or weakness. The maximum stress calculated by the elastic theory is the stress *at a point* on the periphery of the opening. The *average* stress over a *finite length* of a crack must be considered. The larger the opening, the more gradual the stress gradient back into the walls, and therefore the higher the average stress on a given finite length of crack. Therefore, although the maximum stress at a point on the wall may be the same in both a large and a small opening, the larger opening will be less stable, because the given length of crack will be subject to a higher average stress. Fig. 13 shows this condition graphically for a winze 6 ft in diam compared with a shaft 18 ft in diam.

2—The probability for the occurrence of incipient cracks or other weaknesses in the material must be considered. This may be a factor of the volume of material involved, and also the size of the cracks, but a simple comparison of the opening perimeters should give a convenient measure of sufficient accuracy. Thus a larger opening will have a larger perimeter, with more probability of containing weaknesses, and will fail at a lower loading pressure. The difference between the perimeters of a square and a circle of equal greatest dimension is apparently less than the accuracy of the tests. Therefore the graph of Fig. 12 was drawn on the basis of greatest opening dimension alone.

3—The lack of appreciable difference in failure pressures for a square opening and for a round one of the same maximum dimension must be explained. On about two-thirds of the periphery of a square opening in a hydrostatic stress field, the stress is less than that on a round opening. Hence, on that portion of the square opening the probability of failure is less than on a round opening because the stress is lower. Although the stress at the corners of the square may be very high, the probability of a crack of important size occurring *at the corners* is very low, and the lower overall probability of failure raises the stability of the square opening into the same range as that of a circular one.

The Production of Sodium Sulphate From Natural Brines at Monahans, Texas

by William I. Weisman and Ross C. Anderson

THE manufacture of anhydrous sodium sulphate or salt cake from natural deposits in the United States has been in general somewhat of a marginal undertaking. Competition from foreign sources and from large quantities of byproduct sodium sulphate produced domestically in the manufacture of hydrochloric acid and other chemicals has existed and continues.

For example, most of the sodium sulphate produced is a byproduct or co-product in the manufacture of hydrochloric acid through the reaction of sodium chloride with sulphuric acid. In recent years, many manufacturers of rayon have installed equipment to recover sodium sulphate from waste spin bath liquors; today this is an important source.

Before World War II large quantities of sodium sulphate were imported from Germany. In 1949 imported material from Europe again appeared on the domestic market. Natural sodium sulphate from Canada in substantial quantities also enters the United States markets.

Despite this kind of competition, numerous attempts have been made to exploit various natural deposits of sodium sulphate in this country, but only a very few of these have survived economically over a period of years. One of these few operations is the plant of the Ozark-Mahoning Co. located 13 miles south of Monahans in West Texas. Several factors contributing to the successful life of this plant may be summarized as follows:

1—Geographical location. Monahans is reasonably close, freightwise, to the Kraft paper mills in Texas, Arkansas, and Louisiana; the Kraft paper industry is the greatest consumer of sodium sulphate in the United States.

2—Availability of natural gas as low cost fuel. Proximity of the natural gas fields of West Texas has been a tremendous asset, as the availability of low-cost natural gas is to all industry throughout the Southwest.

3—The nature of the deposit. The occurrence of sodium sulphate brines in southeastern New Mexico and West Texas has been very well described by Lang,[1] who writes that the brines are found in the Castile formation of the Delaware basin. Here weathering has altered the anhydrite so that a relatively porous gypsiferous zone overlies a dense impervious mass of anhydrite. This porous zone provides traps where percolating ground waters that have picked up soluble salts may lodge. These traps or pockets are the natural brine reservoirs exploited at Monahans. Although several hundred wells have been drilled, currently some 25 wells serve to supply brine to the plant. All are within 1½ miles of the plant and are conveniently tied together by an electric power system serving electric motors driving the pumps.

Having the raw material in the form of a brine which can be pumped from shallow wells makes possible much simpler and more efficient handling than if it were in form of solids.

By contrast, other deposits of sodium sulphate, such as those in Arizona, Nevada, and North Dakota, are in the form of the solid minerals, thenardite and mirabilite, which present somewhat more of a mining and mineral dressing problem.[2] The largest producer of sodium sulphate from natural sources in the United States is at Searles Lake, Cal., and there a brine also is utilized.

4—Water. Substantial quantities are needed for cooling towers and for operation of gas engines. An area underlain with brine is not a promising source of fresh water, but fortunately, after a long search, an adequate supply was found nearly two miles from the plant.

It may be appropriate to discuss briefly the grades of sodium sulphate offered on the market. Salt cake is the name usually applied to the grade of sodium sulphate used by the Kraft paper industry. It may be a low analysis byproduct, 95 to 97 pct sodium sulphate, with as much as 1½ to 2 pct residual acid, or it may be a natural product. Usually salt cake is considered a low grade product, but a great deal of a higher grade of material is marketed under this name. The specifications for glassmakers' salt cake are somewhat higher than those of the paper industry, usually requiring 98 pct sodium sulphate.

Technical anhydrous sodium sulphate is a high grade material and usually exceeds 99 pct sodium sulphate. It finds the biggest market in the textile industry and is used as a builder in some synthetic detergents.

Glauber's salt, $Na_2SO_4 \cdot 10H_2O$, is usually of high purity. Preferred for some uses, it normally has been recrystallized from an anhydrous salt.

A unique manufacturing process has been developed at Monahans. This process results in the production of an exceptionally high grade of salt cake, and qualifies for nearly all uses, including many which specify the technical anhydrous grade. All of the finished product, which is very white, passes a 10-mesh U. S. Standard screen, and is retained on a 200-mesh U. S. Standard screen. It is over 99 pct Na_2SO_4 with main impurities being sodium chloride and magnesium sulphate. Iron content is less than 0.01 pct.

As mentioned, the raw material at Monahans is a brine drawn from wells. Attention was first attracted to this location because a so-called alkali

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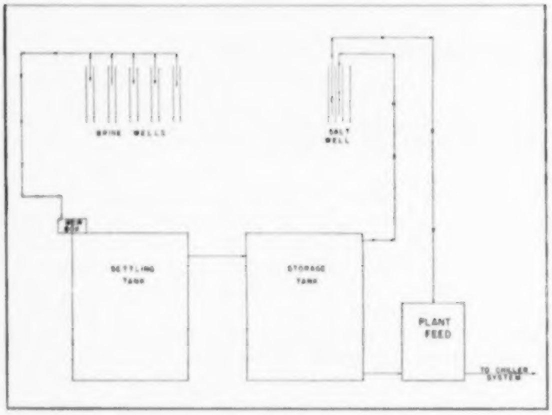

Fig. 1—Flowsheet illustrating handling of brine.

lake here was noted to be rich in sodium sulphate. In 1931 and 1932 investigation and drilling around the shore of the lake and in the general area revealed the existence of the brines. These offered a raw material of uniform nature in comparison to the lake brine, which varies greatly in concentration depending on seasonal factors of rain or drought.

Nevertheless, the lake may be utilized to a certain extent in processing the brine. Actually, two different brines are drawn from two different levels in the wells. At a depth of 60 ft there occurs an abundant supply of weak brine, 1.160 sp gr, and 7 to 8 pct sodium sulphate. In summer brine from this level is pumped onto the lake, where it is concentrated by solar evaporation to 1.180 sp gr, or about 10 pct sodium sulphate. Brine at this concentration is excellent as feed to the plant. Pumping of brine is controlled to maintain a constant level in the lake.

At a depth of 90 ft a strong brine of 1.180 to 1.20 sp gr is found. This brine is used directly as plant feed in winter and supplements the lake in summer. The deeper level has never produced so abundantly as the shallower levels; hence every effort is made to conserve the stronger brine. This brine averages 11.2 pct sodium sulphate with a typical analysis showing 6 pct chloride, 8.2 pct sulphate, 1.8 pct magnesium, and sufficient sodium to satisfy stoichiometric requirements. Traces of other elements are present.

The strong brine wells are pumped at rates of 3 to 18 gal per min. More rapid pumping will cause the brine strength to decrease. The optimum rate of pumping for a given well is determined from experience. Occasionally one of these wells is abandoned because the brine becomes dilute. It is uneconomical to continue to utilize weak brine from the 90-ft level when there is ample weak brine at shallower level.

Pumps are F. E. Meyers pump jacks with a 12-in. stroke, driven by 840-rpm, 2-hp, 220-v motors. The process used in the plant consists of two principal steps. The first is to cool the brine to crystallize out Glauber's salt, which is 56 pct water and 44 pct sodium sulphate. The second step is to melt the Glauber's salt, evaporate the water of hydration, and dry to anhydrous sodium sulphate.

It would be impossible to dehydrate the brine directly because, aside from the considerable fuel cost for evaporation, magnesium sulphate and sodium chloride are present in substantial quantities and a mixed salt would be obtained. Glauber's salt has a very steep solubility curve, and excellent recovery is made by the crystallization method used.

As a matter of fact, sodium chloride is added intentionally to the raw brine before chilling; the result is a further decrease in the solubility of the Glauber's salt. This is accomplished by pumping about one-third of the raw brine into a deposit of

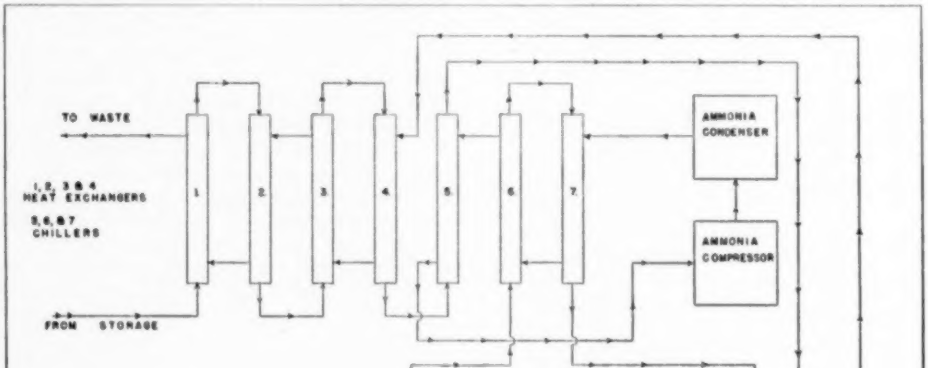

Fig. 2—Glauber's salt crystallization system in chiller plant.

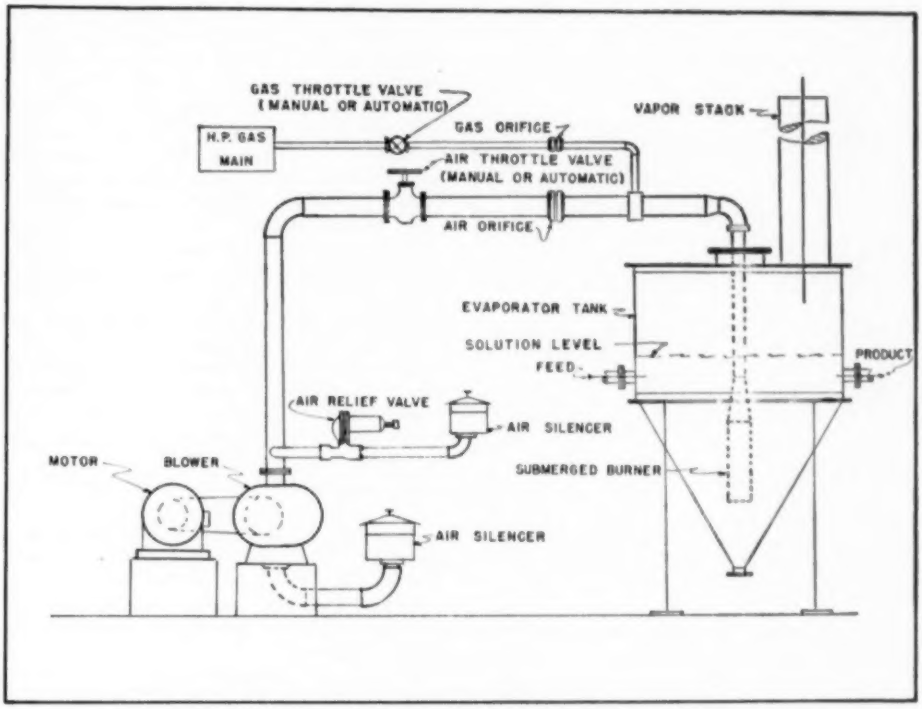

Fig. 3—Schematic sketch of typical submerged combustion installation.

halite, which saturates it with sodium chloride. The halite deposit lies between 1000 and 1500 ft and only one salt well is used, the brine being introduced down the annular space between the tubing and well casing and a chloride saturated brine being pumped up through the tubing.

Fig. 1 is a flowsheet illustrating the handling of the brine. The combined flow from the wells and the lake, or solely from the wells, as the case may be, passes through a weir box which provides a rough approximation of flow and gives the pumper some warning if any of the wells have stopped pumping. From the weir box the brine goes to a settling tank where insoluble material settles out. The overflow goes to a storage tank. Both are wooden tanks 30 ft in diam by 8 ft high. From the storage tank brine is drawn continuously for pumping through the salt well and returned to the system via the weir box. Plant feed brine is withdrawn continuously from the storage tank which serves as a surge tank in case of interruption of flow from the wells.

The plant feed brine is maintained at 1.22 sp gr and contains about 11 to 12 pct sodium sulphate and 12 to 15 pct sodium chloride. The percentage of sodium chloride in the brine has been increased substantially as the original brine contained 9 to 10 pct sodium chloride. In a typical day's operation 43 tons of sodium chloride will be removed from the salt well, but none of it is recovered; all of it after serving its purpose of decreasing the solubility of the Glauber's salt is discarded in the tailing liquor. The plant has a rated capacity of 100 tons of anhydrous sodium sulphate per 24-hr day, and the corresponding flow of brine to the plant is 150 gal per min. Recovery is about 1 lb sodium sulphate per gal of brine passing through the plant. When brine from the lakes reaches a concentration equaling 1.22 sp gr the use of the salt well is discontinued.

Hydrometers are located at several points within this system to enable manual control of the concentration of the plant feed brine. Brine strength is not extremely critical and slight variations have no serious effect on plant operations.

Fig. 2 is a flow diagram of the Glauber's salt crystallization system which functions in the chiller plant. About 85 pct of the sodium sulphate in the brine can be recovered as Glauber's salt by cooling the brine from 90° to 15° to 20°F. To cool 150 gpm of brine and to crystallize the Glauber's salt requires substantially 500 tons of refrigeration. About 300 tons are provided by an ammonia refrigeration system and the remaining 200 tons are recovered through heat exchange from the waste liquor before it is discharged. Ammonia compressors are of Worthington Pump Co. manufacture. Three 100-ton units are driven by 200-hp Cooper-Bessemer gas engines and one 285-ton unit is driven by a 400-hp gas engine.

Three ammonia chillers and four heat exchangers or recuperators are in use. Chillers and recuperators built by Worthington Pump Co. are similar to the chillers used in petroleum refineries to dewax lubricating oil. Each unit consists of twelve insulated tubes 34 ft long. There is an 8-in. diam outer tube with a 6-in. tube on the inside. The brine flows through the inner tube, which is equipped with rotating scraper ribbons to prevent any precipitating Glauber's salt from building up and adhering to the wall. Flow of brine is countercurrent to the cold mother liquor in the annular space and the four recuperators operate in series. Ammonia is the refrigerant in the annular space of the chillers. The entering brine, which may be as high as 90°F in summer and as low as 60°F in winter, passes through the recuperators, and one ammonia chiller is thereby cooled to 35° or 40°F. The brine is then discharged to the first-stage thickening tank, a conventional Dorr thickener of 20-ft diam.

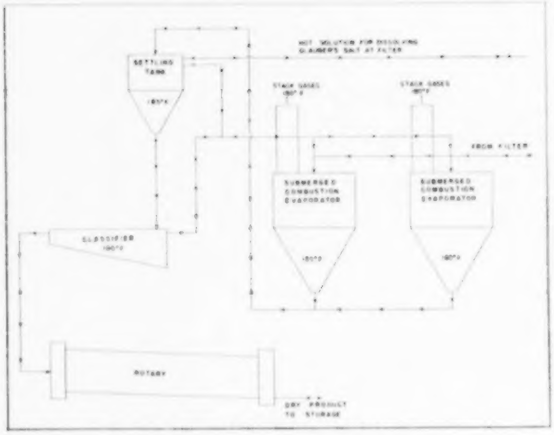

Fig. 4—Flowsheet of drying plant.

The thickened slurry goes to a rotary vacuum filter where the Glauber's salt crystals are filtered and washed. The plant is equipped with both a Swenson and an Oliver filter, but only one is used at a time.

The filtrate and washwater from the filter are combined with the overflow from the first-stage thickener and thence pass through two more ammonia chillers to cool to 15° or 20°F so that additional Glauber's salt will be crystallized. The slurry goes to the second-stage thickener, which is a 26-ft Dorr. The thickened underflow mixes with the underflow from the first-stage thickener and goes to the filter. The overflow is the waste mother liquor at about 20°F and, as stated, is the heat-exchange medium in the recuperators. It is discharged finally to waste at a temperature of 55° to 75°F, depending on the time of year and temperature of the incoming brine. This warm mother liquor is used to flush out and wash out any parts of the system that may have become salted up.

A question often arises regarding the choice of surface cooling instead of vacuum cooling, which is commonly used in similar crystallization problems. Actually, vacuum cooling is efficient only to a terminal temperature of about 40°F, and it is impossible to cool below 32°F by this method. To achieve a good recovery of Glauber's salt it is necessary to chill to about 20°F. Therefore the system in use was chosen.

The drying plant, where the water of crystallization is removed from the Glauber's salt, has been described in detail;[3] however, some time is devoted to it here, as it is probably the first commercial application of submerged combustion evaporation.

Sodium sulphate has an inverted solubility curve, which means less solubility at higher temperatures. For this reason it deposits an extremely adherent scale on heat-transfer surfaces such as the tubes of a vacuum evaporator. Operation of this type of evaporator, in which heat is transferred through a tube, becomes very difficult and inefficient because of the rapid build-up of scale and the consequent loss of heat transfer. As a result there are frequent and expensive shutdowns for cleaning out scale.

When operations began at Monahans in 1933 various schemes were tried to effect the dehydration of the Glauber's salt, but at that time none was satisfactory. Rotary drum driers proved impractical because of the severe scaling. For a time the dehydration was carried out in a counter-current fired rotary kiln. This operation was difficult and inefficient and was replaced by the first commercial submerged combustion evaporator in 1934. The evaporator removes most of the water with final drying of the salt taking place in the rotary kiln.

In the submerged combustion evaporator a gaseous fuel is burned beneath the surface of the liquid, in this case the solution of sodium sulphate, and the products of combustion bubble up through the solution. The burner is submerged at sufficient depth to assure complete and efficient transfer of heat from the products of combustion to the solution. Scaling cannot interfere with this heat transfer, as the heat flows directly from gas to liquid, and rapid heat transfer is assured because the gas is ejected as many bubbles, presenting a tremendous surface area. Furthermore the driving force, which is the temperature difference, is very great, as the flame temperature is around 2600°F.

Fig. 3 shows a typical submerged combustion evaporator. Combustion air is compressed to a pressure of about 7 psi gage and mixed with the proper quantity of gas at equal or slightly higher pressure. The mixture is introduced to the submerged burner and ignition is initiated by a resistance wire heated to incandescence. The flame maintains itself without the presence of any refractory. The controls shown here are manual, as are the controls at Monahans. However, submerged combustion units have been designed with fully automatic control.

The burners used at Monahans are made of mild steel. Some scaling occurs on the tip of the burner, and as it builds up it flakes off, together with a little metal. This action causes sufficient damage to necessitate frequent repair and replacement of burners, but burner cost per ton of production is only a few cents. Some alloys give longer life, but not enough to warrant the extra cost. In other solutions the use of alloys may be desirable, and a graphite burner has been developed for use in acids and corrosive solutions.

Fig. 4 is a flowsheet of the drying plant. About 70 pct of the total water load is evaporated in the two submerged combustion units, or about 110 tons per 24 hr. A dilute slurry of 4 to 5 pct precipitated solids is drawn continuously from the evaporators to a settling tank. The underflow of 20 pct precipitated solids, or 50 pct Na_2SO_4, is further thickened in an Akins type of classifier to 55 pct precipitated solids or 70 pct Na_2SO_4. The balance of the water is driven off in the rotary kiln and the dry salt goes to storage for bagging or shipment in bulk.

Instead of thickening the slurry before feeding the salt to the drier, it would be quite feasible to use a centrifuge or filter which could reduce the moisture content of the salt to 4 or 5 pct before drying. Because the submerged combustion evaporator is much more efficient for evaporation of water than the rotary drier, it may seem that substantial savings of fuel could be effected. Actually, however, the price of gas is such that the savings in fuel cost would be offset by the additional power required for a filter or centrifuge.

The overflows from both the settling tank and the classifier are returned to the submerged combustion evaporator, except for a portion from the settling tank which is used to dissolve Glauber's salt at the filter and introduce solution to the evaporators.

The evaporator tanks are made of type 304 stainless steel with highly polished interior surfaces.

After 15 years of service they show no signs of deterioration. Much of the chilling plant has been rehabilitated recently, so that the plant is in a position to meet increased demands now being made on all producers of mineral products.

Each submerged combustion unit burns 170 cu ft of natural gas per min. Combustion air is supplied to each burner by Roots-Connersville positive pressure blowers driven by 200-hp gas engines.

The plant now has two separate chilling systems, similar to the one described, and interconnected. This means that the same rate of production can be maintained with a weaker brine feed and use of the lake for solar concentration can be eliminated if desirable. If additional production is desired, however, feeding strong brine can increase the production to about 150 tons of anhydrous sodium sulphate per day. Ample filter and evaporation capacity is available in the existing equipment to handle this tonnage, and present indications are that the plant will be called on for maximum production.

In summary, the Monahans sodium sulphate operation is unique in representing a fusion of several favorable factors that have been fortified with new ingenious developments in processing to yield a top quality product in an industry frequently witnessing failure in wrestling with natural deposits.

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Technical Note

Quantitative Bubble Pick-Up Methods

by S. C. Sun and R. C. Troxell

EFFORTS to obtain definite quantitative data when employing the currently used bubble pick-up method[1, 2] as a pre-flotation investigation tool led to the adoption of the *magnifying mirror method* and the microscopic method modifications. Quantitative data obtained by these two methods were converted into bubble pick-up indices and bubble pick-up coefficients. Rough correlations were established between the calculated indices and the results of contact angle testing as well as actual flotation under the same conditions.

The basic steps of the magnifying mirror method are as follows: 1—mixing 200 ml distilled water, sized and cleaned mineral particles of approximately 0.5 g, pH regulator, and flotation reagents in a glass beaker; 2—swirling the mineral particles to the center of the beaker to leave a clear circumference; 3—pressing an air bubble against the mineral particles for about 1 sec, and then moving the mineralized air bubble to the clear part of the beaker; and 4—dropping all the attached mineral particles of the air bubble by tapping the bubble holder, and counting the number of the dropped particles by looking at the magnified image in the mirror positioned beneath the beaker. This method is impractical for mineral particles which are transparent in water or finer than 200 mesh.

The microscopic method is essentially similar to the above described magnifying mirror method, with the exception that the mineralized air bubble was drawn up into the bubble holder and subsequently discharged onto an externally placed watch glass, where the number of pick-up particles was counted with the aid of a microscope or hand lens. Drying was necessary when the discharged liquid interfered with the counting.

The results of the bubble pick-up tests can be simply expressed in the form of the actual count. However, this can be misleading because the number of pick-up particles varies not only with the size of bubble but also with the size of particles. This short-coming is largely remedied by the use of the bubble pick-up index, as calculated from the following equation:

$$PI = (p_a/p)\ 100 \qquad [1]$$

in which PI and p_a are respectively the pick-up index and the number of pick-up particles under any test conditions, and p is the maximum number of pick-up particles of the same mineral under optimum flotation conditions, with the same size holder and particle. The *bubble pick-up coefficient* is a term coined to denote the difference between the bubble pick-up index of mineral particles in a liquid and that of the same mineral-liquid mixture plus a chemical reagent at the same pH value. For example, the difference between the pick-up index of a naturally floatable bituminous coal in a solution of water and kerosene and that of the same coal in water alone is a bubble pick-up coefficient.

Compared with the contact angle method, the quantitative bubble pick-up method is inferior in accuracy but superior in sensitivity. Other advantages of the bubble pick-up method are that the apparatus can be easily assembled, the mineral sample is conventionally prepared, and the test procedure is relatively simple.

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The Blending of Western Coals for Production Of Metallurgical Coke

by John D. Price

COAL blending, in the preparation of coal before coke making, is so commonly practiced as to be almost universal. But the reasons underlying this practice, the benefits resulting from it, and the materials used in blending vary widely. This paper will outline the various phases of the subject and present information which may be correlated with work that has been done elsewhere. It will deal entirely with work done on the high-volatile coking coals of the western part of the United States, special emphasis being given to the coals of Colorado and Utah.

A survey[1] of the 86 coke plants in active operation in the United States during 1949 indicates that only 9 plants, or 10.5 pct of the total, charged one single rank of coal into their ovens, while the remaining 89.5 pct made use of blending in some form. This report indicated that of these total plants 5 used straight high-volatile coal, 4 used straight medium-volatile coal, 47 used blends of high and low-volatile coals, 25 used blends of high, medium, and low-volatile coals, 2 used blends of high and medium-volatile coals, 3 used blends of medium and low-volatile coals.

The fact that certain plants operated on a single kind of coal should not be interpreted to mean that no blending was practiced there, for invariably such plants secure their coal from more than one source and in the interest of uniformity do blend the coals as received.

The general term *coal blending* covers two fields, the first of which is the mechanical mixing of a number of coals to secure uniformity. Often it is found necessary to secure coal for coke production from a number of different mines; these coals, though of the same general type or rank, may differ in their chemical composition or in the physical qualities they impart to coke made from them. Again, it is not unusual to find that coal from different sections of the same mine may show variations in quality. Under such conditions it may be necessary, in the interest of a uniform final product, to introduce a system of blending bins, a bedding yard, or other mechanical methods of securing a uniform mixture. Unfortunately this form of blending has received very little attention up to the present time; it has not received the consideration its value merits.

The second type of blending, while also for the purpose of coke improvement, deals more particularly with the use of a blending agent differing in character from the base coal; it is this form of blending that will be discussed here. To consider only the western coals, for blending may be found necessary for other reasons with other coals, blending has been practiced experimentally or commercially under the following conditions:

1—When a single coal or mixture of coals of the same rank does not produce a satisfactory coke. For example, a high-volatile coal when used alone is likely to contract when coked so that a comparatively weak coke is formed. Or, if of very low rank, the coal may be deficient in the necessary bitumens required for good coke production.

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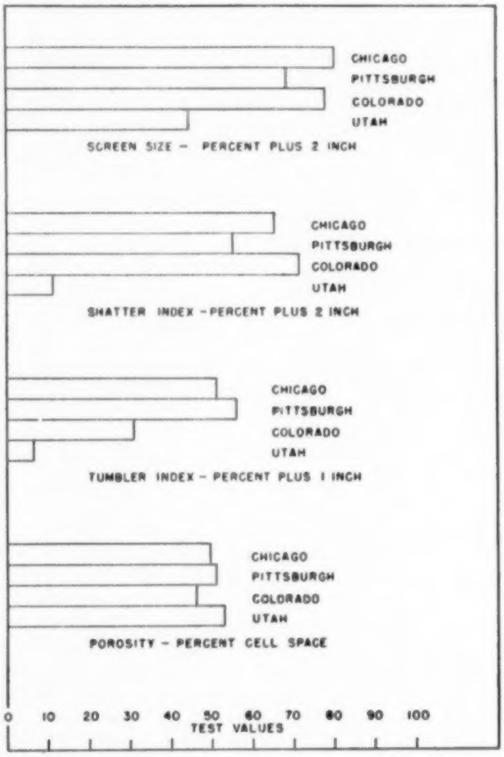

Fig. 1—Comparison of eastern and western cokes.

2—When a product of some special quality is required, for example, when a plant ordinarily producing blast furnace coke must operate at slow coking rate to produce a high-grade foundry coke. Under this condition the reduced daily production of all products which accompanies slow coking time may be undesirable, and the use of some blending agent to increase the size of coke made at faster coking rates may be necessary.

3—When greater yield of coke or its coproducts is needed. Depending upon economic values of the products it may be found desirable to increase the yield of one or the other.

4—When supply of a particular coal must be used, either to protect reserves of high quality coking coal or to utilize a surplus or inferior product not otherwise usable.

Many materials have been used for blending purposes, the exact agent to be used depending both upon the condition to be corrected and the nature of the base coal. No universal blending agent that can

Table I. Analyses of Colorado and New Mexico High-Volatile Coals

Mine	Frederick	Morley	Koehler	Crested Butte	Kebler
Classification	Coking	Coking	Light-Coking	Semi-Coking	Non-Coking
Kind	Washed	Washed	Washed	Washed	Washed
Moisture*	2.6	3.8	3.9	7.3	6.2
Volatile matter**	31.0	30.0	37.3	39.6	37.6
Fixed carbon	57.5	55.7	48.4	55.4	51.7
Ash	11.5	14.3	14.3	5.0	10.7
Phosphorus	0.039	0.001	0.001	0.014	0.001
Carbon†	84.61	83.72	82.09	79.78	77.62
Hydrogen	5.74	5.80	6.00	5.63	5.41
Oxygen	7.32	7.86	9.44	12.46	14.83
Nitrogen	1.80	1.80	1.67	1.68	1.48
Sulphur	0.53	0.82	0.80	0.45	0.66
Free swelling index	8.5	7.0	3.5	1.5	0.0
Agglut. index 10 to 1	12.0	10.1	7.7		
Agglut. index 15 to 1	7.7‡	5.8‡	3.4‡		
Source of information	Ref. 8	Ref. 8	Ref. 8	Ref. 8	Ref. 8

* As-received basis, on raw coal from mine
** Moisture-free basis
† Moisture and ash-free basis
‡ Estimated value

be used to improve the coke made from all coals is available. The nearest approach to this will be found in the use of a low or medium-volatile coal as practiced at 75 pct of the plants in the United States. But even in this event each plant must work out the proper type and quality of coal to use and must consider carefully the economics of the practice. In effect, coal blending is an individual plant problem and must be worked out by each plant to secure best results.

At the western coke plants, the following blending agents have been used. Of these the first three have been used on a commercial scale and the last three only in more or less extensive experimental tests: 1—coking coals of lower volatile content and of different expansion properties, 2—non-coking high volatile coals, 3—coal tar pitch, 325° to 350°F melting point, 4—low temperature char, 5—coke breeze, 6—petroleum coke breeze.

Substitute Materials and Procedures

At times it may be found that the blending agent which appears to be most desirable to correct a certain condition is not available in that particular locality and can be secured only at excessive cost. In such event a substitute material can often be secured or produced. Thus while anthracite fines are desirable for use in increasing the size of coke pieces, fine coke breeze can be used in its place with nearly as good results. Again, low-volatile coking coals are not always economically available, and in such localities low temperature char produced from local high-volatile coals presents a very acceptable alternate. The use of such materials will be touched upon later.

While in most cases not as effective as the use of proper blending agents, certain alternate procedures have been practiced at many plants with gratifying results. These will be merely listed here:

1—Control of carbonizing temperature has an effect on yields of coke and coproducts and on the physical properties of the coke[2,3] (p. 217 of ref. 2).

2—Degree of pulverization has an effect on the physical structure of the coke[2,4] (p. 214 of ref. 2).

3—Control of bulk density of the charge has an effect on the expansion pressure exerted by the coal and on the quality of the coke[2] (p. 93 of ref. 2).

4—Control of moisture content of the charge has an effect on bulk density and consequently on the properties of coke[2,5] (p. 93 of ref. 2 and p. 136 of ref. 5).

5—Washing of the coal prior to coking has in many cases been proved to benefit both the chemical and physical properties of the resultant coke[6] (p. 32 of ref. 6).

6—The method of charging ovens has been found to be of importance in securing uniformity of bulk density and quality of coke throughout the whole oven[5] (p. 96 of ref. 5).

Table II. Analysis of Utah High-Volatile Coals

Mine	Sunnyside	Columbia	Book Cliffs	Castlegate	Huntington
Classification	Light Coking	Light Coking	Semi-Coking	Semi-Coking	Non-Coking
Kind	Raw	Raw	Raw	Raw	Raw
Moisture*	5.8	5.3	9.0	4.1	3.4
Volatile matter**	40.8	40.1	39.2	41.6	43.8
Fixed carbon	53.4	54.4	55.2	52.3	49.6
Ash	5.8	5.5	5.6	6.1	6.6
Phosphorus			0.019	0.012	0.001
Carbon†			79.13	79.66	79.56
Hydrogen			5.74	5.76	5.89
Oxygen	8.7	10.0	12.68	12.68	12.63
Nitrogen			1.59	1.52	1.49
Sulphur	1.2	0.8	0.86	0.38	0.43
Free swelling index	4.8	4.5	3.0	2.0	1.0
Agglut. index 10 to 1	8.1‡	7.0‡	3.3	2.6	
Agglut. index 15 to 1	3.9	3.0	1.0‡	0.5‡	
Source of information	Ref. 12	Ref. 12	Ref. 13	Ref. 13	Ref. 13

* As-received basis
** Moisture-free basis
† Moisture and ash-free basis
‡ Estimated value

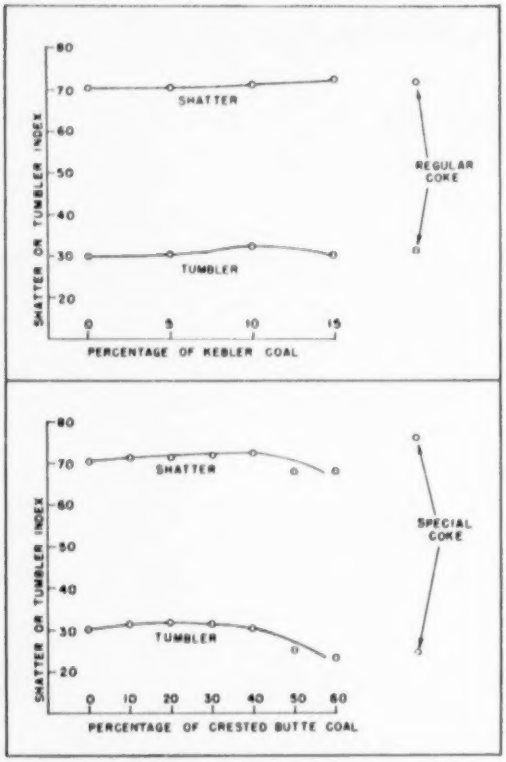

Fig. 2—Effect of blending coals of similar rank.

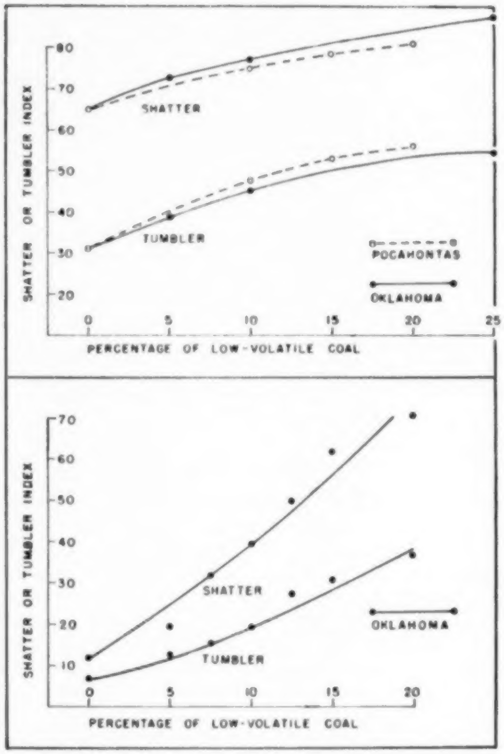

Fig. 3—Effect of blending high and low-volatile coking coals.

Very few high-volatile coking coals found in Colorado and Utah can approach the coals of the East as to quality of coke that can be produced from them. In fact, some of these western coals used regularly in the production of coke are so deficient in coking properties that they would be classed as non-coking by an eastern operator. They differ in the main from eastern coals in having been formed in the Cretaceous period and are usually of lower rank and higher oxygen content. Analyses of five representative coals from the Raton Mesa district of Colorado and New Mexico are shown in Table I. Nearly all these coals are high in ash and require washing; the analyses of washed coal samples only are shown. Because Utah coals, on the other hand, are often low in ash, washing is of minor value[7] (p. 9 of ref. 7). Table I shows analyses of five coals from Utah.

No low-volatile coking coals of a nature similar to Pocahontas are found in the Rocky Mountain states; the nearest source of supply of this kind of coal is in Oklahoma and Arkansas. On the other hand, Kebler coal, a high-volatile, high-oxygen, non-coking coal, the analysis of which has been given in Table I, has been used continuously in small amounts, and char, prepared by low-temperature partial devolatilization of high volatile coal, has been used experimentally. Analyses of such blending agents as have been used with western coals are shown in Table III.

Table III. Analysis of Low and Medium-Volatile Coals and Char

Mine	Carswell*	Philpott	Bokoshe	Stigler	Char
Classification	Low-volatile coking	Low-volatile coking	Low-volatile coking	Medium-volatile coking	
State	West Virginia	Arkansas	Oklahoma	Oklahoma	Colorado
Kind	Raw	Raw	Raw	Raw	
Moisture**		5.3	2.8		
Volatile matter†	17.1	15.9	18.3	24.6	17.4 to 19.1
Fixed carbon	78.0	77.4	66.8	71.1	72.1 to 70.4
Ash	4.9	6.7	14.9	4.3	10.5
Phosphorus		0.014		0.007	
Carbon‡	91.2	89.33	90.9	87.43	
Hydrogen	4.6	4.58	4.4	5.32	
Oxygen	2.1	2.39	2.0	4.38	
Nitrogen	1.3	2.11	1.8	2.13	
Sulphur	0.8	0.93	0.7	0.74	0.6
Free swelling index	9.0+	9.0+		9.0+	0.0
Agglut. index 10 to 1	20.0+	20.0+		17.7	0.0
Agglut. index 15 to 1	12.0§	12.0§		10.0§	0.0
Source of information	Ref. 5	Ref. 8	Ref. 8	Ref. 8	Ref. 11

* Pocahontas field
** As-received basis
† Moisture-free basis
‡ Moisture and ash-free basis
§ Estimated value

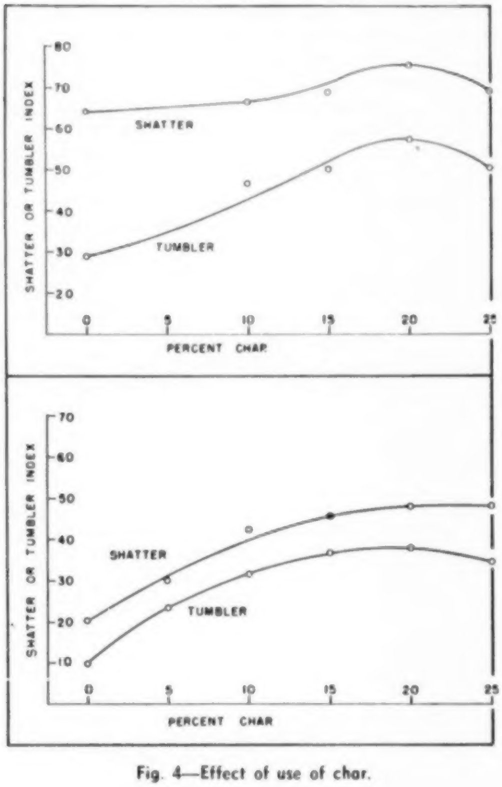

Fig. 4—Effect of use of char.

A comparison of cokes made from eastern and western coals is of value. This is given in Fig. 1, where the various physical properties from representative samples of coke from the Chicago and Pittsburgh districts[5] (p. 132 of ref. 5) are shown for comparison with similar data on cokes made of coals from Colorado[8] and Utah[7] (p. 18 of ref. 7). It must be pointed out here that this comparison is intended to emphasize the need for blending with the western coals, for the eastern cokes were made from coal mixtures containing 20 pct Pocahontas, while coke from western coals was made from straight high-volatile coals. Since the need for blending has been established, and since the examples given here have shown the greatest need for improvement to exist in the shatter and tumbler values, these indices only will be used in a discussion of benefits. The tests should prove to be acceptable criteria of the physical properties of coke, since their cumulative value has been shown to be 80 pct of the total value of coke from a physical property viewpoint at the blast furnaces.[9]

Blends of Colorado High-Volatile Coking Coal with Other High-Volatile Coals of Lower Rank: As shown in Table I, a large number of high-volatile coals of different ranks and coking properties are found in Colorado and New Mexico. Of these Frederick is the only one which possesses properties equal to those of the better coking coals of the East. This coal also has the best washability of any of the coking coals, but alone makes a very dense and fingery coke. Two high-oxygen coals are also found in Colorado, Crested Butte having only slight coking properties and Kebler having none. Between Frederick on the one extreme and these high oxygen coals on the other are found a large number of coals of various intermediate oxygen content and coking properties. Since both Crested Butte and Kebler coals are desirable because of their ash content, an effort was made to utilize them in the coking mixture; the results of such tests are shown in Fig. 2[5] (pp. 20-25 of ref. 5). It will be noted that some slight improvement in both shatter and tumbler indices are shown up to 10 pct for Kebler and 30 pct for Crested Butte. Although not shown in Fig. 2, noticeable improvement resulted in the size of the coke prisms from the use of either of these high oxygen coals.[8] Since the high cost of Crested Butte coal made its use prohibitive, the use of 10 pct Kebler was adopted as standard practice, and since Frederick coal was not available in sufficient quantity, the use of coals similar to Morley and Koehler, see Table I, was necessary. The physical properties of coke made from a blend of 40 pct Frederick, 50 pct miscellaneous coking coals, and 10 pct Kebler is shown as *Regular Coke.*[8] A *Special Coke* mixture, containing 60 pct Frederick and 40 pct Crested Butte was produced occasionally to make a coke of comparatively low ash content; the physical properties of such coke are also indicated in Fig. 2.

Blends of Colorado and Utah High-Volatile Coking Coals with Low-Volatile Coking Coals: Since the standard blend used in 75 pct of the coke plants in this country consists of various proportions of high- and low-volatile coking coals,[1] it seems reasonable to expect that such blending practice should prove to be of value when applied to the western coals. The effect of this is shown in Fig. 3. In the upper section of this chart it will be noted that Colorado high-volatile coal blended with either Pocahontas or Oklahoma low-volatile coal gave similar results.[8] Utah coal when blended with Oklahoma low-volatile coal also shows gratifying improvement[7] (p. 18 of

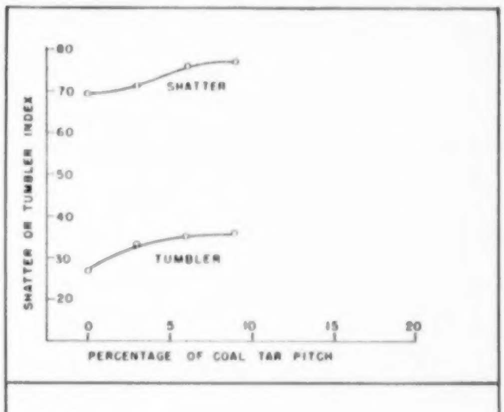

Fig. 5—Effect of use of coal tar pitch.

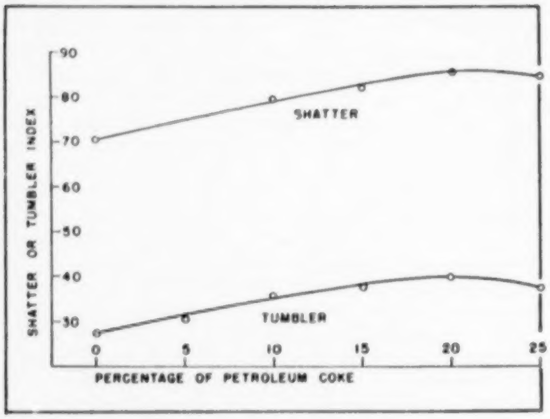

Fig. 6—Effect of use of petroleum coke.

ref. 7) and the result of such blending is shown in the lower section of Fig. 3. One important difference should be noted: with Colorado coal the curves flatten out as the percentage of low-volatile coal is increased, whereas with Utah coal the curve continues to rise. This may be taken as an indication of the deficiency of coke-forming bitumens in Utah as compared with Colorado coals. One additional point is of interest and also serves to illustrate the difference between the Colorado and the Utah coals: the shatter and tumbler values as secured with about 20 pct low-volatile coal additions to Utah coal show physical test values very little higher than those for Colorado coal with no low-volatile coal added.

Blending High-Volatile Coals with Char: It may be found that low-volatile coking coals are not available for economical use as blending agents. In such cases char may be found to be a very desirable substitute. This has been demonstrated at several plants using western coals, and Colorado and Utah coals respond well to the use of this material. In the preparation of char by the partial devolatilization of high-volatile coals at low temperatures, it has been demonstrated that the char must be in effect *tailor-made* to yield optimum results with each base coal used. It has been found that in preparing char the following are important:[10] 1—the coal from which the char is made, 2—the amount of char to be used in the final mixture, 3—the degree of pulverization of the char, 4—the average volatile content of the char and its variation from this average, and 5—the physical properties of the char, with particular reference to its surface.

When char is properly made and used, its effects are similar to those achieved by the use of low-volatile coals. Results obtained from tests of char used with Colorado coal[11] (p. 7 of ref. 11) and with Utah coal[7] (p. 30 of ref. 7) are shown in Fig. 4.

Blending with Coal Tar Pitch: If, as has been previously suggested, Utah coals are deficient in coking bitumens and hence lack fluidity, it would appear that the addition of some bituminous material of highly fluid nature when hot should be of benefit. Coal tar pitch represents such a material, and the effects of its use with Colorado coal[5] (p. 95 of ref. 5) and with Utah coal[7] (p. 14 of ref. 7) are shown in Fig. 5. It will be noted that considerable improvement is indicated in both tumbler and shatter indices. One possible reason for this in connection with the Colorado coals may be found in the fact that certain of the coals from New Mexico which are used in the base high-volatile mixture may be as deficient in bitumens as are the Utah coals.

Blending with Petroleum Coke: The effect of using petroleum coke of about 10 pct volatile matter content in various proportions with Colorado high-volatile coals is shown in the upper section of Fig. 6[5] (p. 92 of ref. 5). A comparison of these curves with those shown for the use of coal tar pitch will indicate considerable similarity. Both these blending agents were considered impractical because of their scarcity.

Blending with Anthracite Fines or Coke Breeze: The use of specially prepared anthracite fines is not uncommon in cases where it is desired to make a coke of size suitable for foundry use yet retain the volume of products which result from fast coking times. Anthracite fines of the proper screen size specifications are not available in the Rocky Mountain states, and fine coke breeze was used as a substitute. It was found that the desired increase in size could be secured and that this was accompanied by an improvement in shatter values, but also by a lowering of resistance to abrasion. The use of breeze also appeared to affect certain combustibility characteristics of the coke to an unsatisfactory extent, and its use was discontinued before any authentic data were secured.

Conclusion

It should be stressed that each coal presents a problem for individual study; no general rule applies to all. Although it has not been mentioned, the matter of cost must not be overlooked; the cost of blending procedures must be measured against the value of benefits secured. Such a study might disclose that the cost of blending is greater than the value of the improvement in coke quality.

References

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[4] R. A. Mott and R. V. Wheeler: *The Quality of Coke*, p. 373. Chapman and Hall, Ltd., London, 1939.

[5] Colorado Fuel and Iron Corp.: *Report on Investigations on Improvement in Coke Quality*, Pueblo, Colo., 1940. Unpublished.

[6] United Nations: *An Enquiry Into the Iron and Steel Industry of Mexico*, p. 32. Published 1951. Distribution restricted.

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[8] Colorado Fuel and Iron Corp.: Unpublished laboratory and operating reports, Pueblo, Colo., 1951.

[9] R. W. Campbell: *Physical Testing of Coke and Correlation with Furnace Operation*. Unpublished paper read at Chicago District Blast Furnace and Coke Association Meeting, 1936.

[10] J. D. Price: *Low Temperature Char as a Substitute for Low Volatile Coal in the Preparation of Metallurgical Coke*. Economic Commission for Latin America, Bogata, 1952. Distribution restricted.

[11] J. D. Price and G. V. Woody: Production and Use of Low-temperature Char as a Substitute for Low-volatile Coal in the Production of High-temperature Coke. *Trans.* AIME (July 1944) **157**, pp. 317-337.

[12] U. S. Bur. Mines: *Coking of Western Coals*. Unpublished report, Denver, Colo., 1943. Published in part in U. S. Bur. Mines *R.I.* 4171.

[13] Colorado Fuel and Iron Corp.: *Analysis of Representative Coals from Western States*. Pueblo, Colo., 1952. Unpublished report.

aime NEWS

Announce Papers For Nova Scotia Meeting

Technical session arrangements are complete for the joint meeting of the Industrial Minerals Div. of the AIME and the Canadian Institute of Mining and Metallurgy with the Mining Society of Nova Scotia, September 8 to 13. Meeting headquarters will be the Keltic Lodge, Ingonish.

The first technical session is scheduled for Wednesday, September 9, starting at 10 am. Papers to be presented are *Industrial Minerals in the Chemical Industry*, by J. R. Donald; *Properties and Application of Iron Blast Furnace Slag*, by J. R. Wallace, N. Weiner, and P. Fedora; and *Geology and Mining of Nova Scotia Gypsum*, by N. R. Goodman and C. B. Hume.

Another technical session is scheduled for Thursday morning. Papers listed for the meeting are: *Petrology and the Industrial Minerals*, by Ian Campbell; *Fluorspar Resources of Newfoundland*, by G. F. Carr; *Present and Proposed Fluorspar Milling, St. Lawrence, Newfoundland and Wilmington, Delaware*, by W. E. Seibert, Jr.; and *Phosphate Rocks as an Economic Source of Fluorine*, by W. H. Hill and K. D. Jacob.

Three papers are scheduled for the third technical session, Friday. They are: *Development of a System of Mine Valuation in New York State for the Purpose of Establishing Assessment Equalization Rates*, by J. J. Prucha and J. R. Thorne; *Some Geologic Features of a Potash Deposit*, by C. L. Jones; and *Importance of Water in Industry*, by J. F. L. Thomas.

Historic Fort Anne at Annapolis Royal, Nova Scotia, is one of the sights awaiting AIME members. The old officers' quarters, built by order of the Duke of Kent, is now a museum.

The last technical session takes place Friday afternoon with the following papers planned for presentation: *Barites Resources of Nova Scotia*, by C. O. Campbell; *Kyanite in Canada*, by V. A. Haw; and *Methods of Proving Worth of Clay and Shale Deposits*, by J. F. McMahon and W. Brownell.

In addition to the technical sessions, there will be four sound films Wednesday evening. Titles of films are: *Photogeology, Bricks From the Sea, Nova Scotia Industries*, and *Sulphur*.

Afternoons will be spent largely on sports, field trips, and sightseeing tours. A series of sports contests, including golf, tennis, archery, darts, lawn bowling, shuffle board, quoits, and ping-pong, are scheduled for Thursday starting at 2:30 pm. In addition, there will be an opportunity for bridge lovers to match cards and wits.

White sails are mirrored in the smooth surface of Halifax's aquatic paradise, The North West Arm. Nova Scotia's shoreline offers almost unlimited aquatic opportunities.

AIME headquarters needs copies of **Ore Deposits of the Western States** and will purchase them at $3.50 per volume.

H. DeWitt Smith Heads 1954 AIME Nominations Slate

For President

H. DeWitt Smith, mining engineer, Newmont Mining Corp., New York, has been nominated for the Presi-

H. DeWITT SMITH

dency of AIME in 1955. Mr. Smith was born at Plantsville, Conn., and graduated from Hartford Public High School. In 1908 he received his Ph.B. from Yale University and his E.M. in 1910. After graduation from college he joined J. E. Spurr, Inc., as an engineer, and worked in the western United States and Mexico. From 1914 to 1917 Mr. Smith was foreman, mine superintendent, and assistant manager, successively, for Kennecott (Alaska) Copper Corp. He then joined the United Verde Copper Co., Jerome, Ariz., as mine superintendent. He later became general superintendent and was sales manager, New York City, from 1927 to 1930. Thereafter Mr. Smith was an engineer in the industrial dept. of the New York Trust Co. for three years. He then became associated with the Newmont Mining Corp. and has been a director since 1930 and a vice-president since 1946. He is also a director and member of the executive committee, Phelps Dodge Corp. and St. Joseph Lead Co. Mr. Smith is managing director of the O'okiep Copper Co., Ltd., and Tsumeb Corp., Ltd. He was adviser, vice-president and executive vice-president, Metals Reserve Co., a subsidiary of RFC, from 1941 to 1944. At this time Mr. Smith had been a Director of AIME, but resigned when he joined Metals Reserve Co. He was also Chairman of the AIME Nominating Committee in 1939. For nine years he has been a member of the Seeley W. Mudd Memorial Fund Committee and is now Chairman. In 1949 Mr. Smith was a Vice-Chairman of the 75th Anniversary Committee, AIME. He is a past president of the Mining & Metallurgical Society of America and also of the Mining Club.

For Vice-Presidents

Theodore Benton Counselman has been nominated as a Vice-President and Director of AIME. Mr. Counselman is manager of FluoSolids sales for The Dorr Co., Stamford, Conn. Columbia University gave him his E.M. degree in 1907 and he then began his career in the test mill of the Inspiration Copper Co. For a few years he was associated with the Cananea Consolidated Copper Co. and from 1912 to 1915 was a metallurgist for the Arizona Copper Co., Ltd. Mr. Counselman was engaged in research work for the Miami Copper Co. for one year and later worked in Minnesota for the Mesabi Iron Co. He became sales engineer with the Celite Products Co., Chicago in 1923. For two years he was Western representative for the Clark Car Co. and then joined Fort Pitt Mine Equipment, Pittsburgh, as sales manager. Mr. Counselman joined The Dorr Co., and has been sales engineer; manager, industrial div.; manager, contract engineering div.; and manager, FluoSolids sales, respectively. Among his inventions are magnetic separators, equipment for manufacturing synthetic rubber, and classifiers. He has written numerous technical articles on handling of flue dust in blast furnace plants, concentration of low grade iron ores, and manufacturing of cement. Mr. Counselman has been active in AIME as Chairman of the Chicago and New York Sections, Blast Furnace and Raw Materials Vice-Chairman, and Iron and Steel Div. Vice-Chairman. He has also served on awards and admission committees. For the last three years he has been Chairman of the National Membership Committee. Mr. Counselman is also a Member of the Executive Committee of the Institute and was named a Director in 1952. He was recently appointed a Vice-President for a one-year term. He holds membership in various professional societies and clubs.

T. B. COUNSELMAN

H. DECKER

Harold Decker, president, Houston Oil Co. of Texas, has been nominated for Vice-President and Director of AIME. Mr. Decker received his college education at Kenyon College, Gambier, Ohio, and the University of Oklahoma. He has also attended the Advanced Management Program of the Harvard University Graduate School of Business Administration. He has been associated with the Skelly Oil Co. and also the Seaboard Oil Co. In 1945 he joined the Pan American Production Co. as assistant manager and was promoted to vice-president and general manager. He resigned from this firm in 1950 to accept the presidency of Houston Oil Co. He is also president of the East Texas Pulp & Paper Co. Mr. Decker is a director of Halliburton Oil Well Cementing Co., Duncan, Okla., and of the South Texas National Bank, Houston. He is also a trustee of Southwest Research Institute and the Institute of Inventive Research, San Antonio. For years he has been active in various oil industry organizations. He

S. S. Clarke, Chairman of the Nominating Committee, has announced nominations for President-Elect, Vice-Presidents, and Directors for 1954. As provided in Art. IX, Sec. 2, of the bylaws, 25 Members or Associate Members may sign and transmit to the Secretary's office prior to Sept. 1 "any complete or partial ticket of nominees," should they wish other candidates to be considered.

In such instance, a letter ballot will be forwarded to all Members in good standing in the United States, Canada, and Mexico, tabulating both the official ballot and any supplementary nominations. If no supplementary nominations are thus received, no letter ballot will be printed, and nominees on the official ballot shall be declared duly elected at the meeting of the Board of Directors or the Executive Committee in November.

holds membership in the API and is a director of the Texas Mid-Continent Oil and Gas Assn. Mr. Decker is a director of the Independent Petroleum Assn. of America and vice-president of the Southeast Texas District. In 1947 he was Chairman of the AIME Gulf Coast Section. In 1948 and 1949 he was a Member of the Petroleum Branch Executive Committee and has been a Director of AIME since 1950.

For Directors

E. C. Babson, who has been nominated a Director of AIME for a three-year term, has been manager of the Canadian div. of Union Oil Co. of California, Calgary, Alberta, since 1951. Born in Los Angeles, he is a graduate of Stanford University, with an A.B. in 1930 and an M.E. in 1933. On graduation in that year he joined the Union Oil Co. of California as a research chemist at Wilmington, Calif. Mr. Babson has held various positions with this company. From 1936 to 1944 he was production engineer at Compton, Los Angeles, and Santa Fe Springs, Calif. For two years he was a lecturer in metallurgy in a war training program at the University of Southern California. Mr. Babson was made district production foreman for Union Oil in 1944. He became staff engineer at Whittier, Calif. in 1948, and in 1949 was appointed assistant chief petroleum engineer, Los Angeles. In 1951 he was acting chief, reserves section, production div. Petroleum Administration for Defense.


E. C. BABSON


G. D. DUB

George D. Dub, assistant to the president, Cyprus Mines Corp., Los Angeles, has been nominated Director of AIME. Mr. Dub received his B.S. from City College, New York, and his E.M. from Columbia School of Mines. From 1912 to 1925 Mr. Dub was employed as a mining engineer by various companies. He has been located in Arizona, Montana, New Jersey, Washington, D. C., Chile, and Peru. In 1926 he was superintendent, general manager for the Pacific Alkali Co. and Pittsburgh Plate Glass Co., Bartlett, Calif. For approximately five years, Mr. Dub was a consulting engineer in Los Angeles and in 1951 joined Cyprus.

Ralph E. Kirk, manager of raw materials, Tennessee Coal & Iron Div., U. S. Steel Corp., Fairfield, Ala., has been nominated as a Director of AIME for a three-year term. Mr. Kirk joined U. S. Steel in 1936 and was appointed to his present position in 1943. Born in Luthersburg, Pa., he is a graduate of Pennsylvania State College. Mr. Kirk was superintendent of the Luzerne Coal & Coke Co. from 1914 to 1916 and then joined the H. C. Frick Coke Co. as superintendent. Mr. Kirk joined the Army during World War I and, after his discharge with the rank of captain, he rejoined the Frick Coke Co. In 1930 he accepted the position of general superintendent, Mahanoy div., Philadelphia & Reading Coal & Iron Co. He holds membership in several professional societies and had been a Chairman of the AIME Coal Div.


R. E. KIRK


C. C. LONG

Carleton C. Long has been nominated for Director of AIME for a three-year term. Dr. Long was Chairman of the Metals Branch Council in 1951, Chairman of the Extractive Metallurgy Div. in 1950, and served as Chairman of the Lead-Zinc Committee in 1948. Born in Boulder, Colo., Dr. Long received his B.S. in chemical engineering from the University of Colorado in 1931. In 1932 he received an M.A. in chemistry from Stanford University and a Ph.D. from Colorado in 1935. He has been associated with the St. Joseph Lead Co. zinc smelting div. since 1935. He started as a research engineer at the Josephtown smelter. In 1937 he was appointed director of the plant research dept., Monaca, Pa., a position he still holds. Dr. Long has made contributions to the literature of the electrothermic winning of zinc and is a member of several professional societies.

Earl Robert Marble, currently manager of the Tacoma smelter, American Smelting & Refining Co., Tacoma, Wash., has been nominated as a Director for three years. A native of Attleboro, Mass., Mr. Marble graduated from Tufts College in 1908. Prior to joining AS&R in 1914, Mr. Marble was a chemist with the Nevada Douglas Copper Co., Ludwig, Nev. His first position with AS&R was that of chemist at Hayden, Ariz. He was promoted to superintendent in 1923 and transferred to El Paso in 1927 as general superintendent. He held the same position at Tacoma until 1947 when he was appointed manager. Mr. Marble is a director of various organizations including the Northwest Mining Assn., Assn. of Washington Industries, and Tacoma Chamber of Commerce.


E. R. MARBLE


P. J. SHENON

Philip J. Shenon has been nominated as an AIME Director for a three-year term. Dr. Shenon was recently named a director of Day Mines, Inc. He is an authority on the Coeur d'Alene silver belt. Dr. Shenon has been in the employ of Day Mines since shortly after the end of World War II as a consulting geologist. He was born at Salmon, Idaho and is a graduate of the Universities of California, Idaho, and Minnesota. Following graduation he was employed as a geologist for the San Francisco Mines of Mexico, Continental Oil Co., and Idaho Bureau of Mines, successively. In 1928 he joined the staff of the Montana School of Mines as an assistant professor of geology. Two years later Dr. Shenon became associated with the U. S. Geological Survey and worked in western mining districts. He then worked in Canada as exploration geologist for the International Nickel Co. of Canada and returned to the United States in 1942 as consulting geologist. During World War II he served in the topographical mapping section of the Army. After his discharge he was named dean of the School of Mines, University of Utah. He resigned in 1952 to set up private consulting practice. Dr. Shenon was Chairman of the AIME Columbia Section in 1947, Chairman of the Mining Geology Committee in 1948, and was first Chairman of the Mining, Geology and Geophysics Div. in 1949 and 1950. Dr. Shenon was active in the formation of the latter division. He was a Member of the Mining Branch Council in 1950 and 1951. He holds membership in professional societies in the United States and Canada.

Juarez, Mexico is just a short trolley ride across the International Bridge from El Paso.

El Paso Program Complete

Plans have been finalized for the AIME Regional Meeting to be held October 28 to November 2 at El Paso. The meeting is being held in connection with the City's International Mining Days celebration.

Program arrangements for the Industrial Minerals Div. and Mining Geology are complete with two field trips scheduled for the last two days of the five day program. One trip will be to the Carlsbad Caverns at Carlsbad, N. Mex., Sunday, November 1. The Carlsbad potash mines are on the schedule for the next day.

Industrial Minerals papers will include such subjects as the industrial minerals of the Navajo area of Arizona; *Bentonite in McKinley County*, by John Allen; and *New Developments in Carlsbad Area*.

International Mining Days is sponsored by the El Paso Chamber of Commerce. This year's celebration is expected to be one of the biggest in the event's history. Mining men from the U. S., Mexico, and Canada are already making plans to attend. Members planning to be at the Regional Meeting should contact T. R. Lind, Chairman, Hotel & Housing Committee, International Mining Days '53, Chamber of Commerce, 310 San Francisco St., El Paso, Texas.

The Hotel Paso Del Norte will be meeting headquarters, but accommodations range from hotels to motels. Rates vary according to the type of accommodations desired.

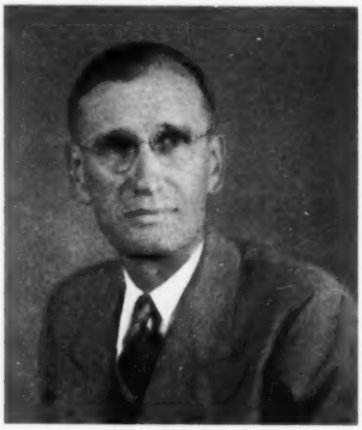
El Paso, called the "sunshine playground of the Border," is close enough to old Mexico to make walking, personal car, or taxi feasible transportation for a short jaunt south of the border.

El Paso Program

Oct. 28		Registration
	Evening	Committee & Directors meetings
Oct. 29		Registration
	Noon	Welcoming luncheon
	pm	First technical session
	Evening	Open
Oct. 30		Registration
	am	Second technical session
	Noon	Group luncheons
	pm	N.M.M.A. (General meeting)
	Evening	A.I.M.E. Banquet
Oct. 31		Ranchero Breakfast
	am	Third technical session
	Noon	Open
	pm	Fourth technical session
	Evening	Suppliers Party
Nov. 1		Field trip—Carlsbad Caverns
Nov. 2		Field trip—Potash Mines

Hibbing Meeting Committee in Full Swing

Preliminary plans for the Fall Meeting of the Minerals Beneficiation Div., August 31 to September 2, at Hibbing, Minn., include an education session, a morning of technical discussion, and a series of field trips. Grover J. Holt is Chairman of the Local Arrangements Committee of the host Minnesota Section. Finance; Housing; Publicity; Registration; Technical; Trips; Transportation and Luncheon; and Entertainment Committees are busily at work setting up the facilities and activities for the meeting.



GROVER J. HOLT

Field trips are scheduled for September 1 and 2. They will cover: Heavy media separation and cyclone plant; sintering plant; taconite plant; stripping and mining operations using belt conveyors, trucks, and draglines; and fusion piercing. One morning will be devoted to technical sessions.

Meeting registration for members will be \$4.00; nonmembers \$6.00; and students will be admitted without charge. Checks must accompany preliminary registration.

The Housing Committee warned those who plan to attend the meeting not to wait to the last minute to make reservations.

James K. Mullins, Coeburn, Va., admires watch awarded to him as outstanding student in coal mining engineering at Virginia Polytechnic Institute. The award is made annually by the Old Timers Club, composed of leading engineers in the country.

A. Fletcher Addresses Missouri Students

AIME President Andrew Fletcher pointed to the advantages offered by Institute membership to a gathering of students on the campus of Missouri School of Mines and Metallurgy.

In his talk to the students, Mr. Fletcher also stressed personal attributes needed by young men entering the mining field. He also compared the U. S. lead and zinc position with that of the rest of the world. Mr. Fletcher predicted a healthy future for the U. S. industry.

Adirondack Section Elects Delegates

Paul Allen was elected to fill the unexpired term of C. DeWitt Smith as Adirondack Section Delegate to the Annual Meeting. Mr. Smith previously requested that a delegate be named in his place in July for an additional two years.

G. Fleck was elected as an alternate delegate to the Annual Meeting and will automatically succeed Mr. Allen as regular delegate in 1956 for a two year term. Both elections took place at the Section's May meeting at Star Lake, N. Y.

H. H. Vogel, Section President, announced that AIME President Andrew Fletcher would address the August meeting of the group scheduled for Gouverneur, N. Y. The September meeting has been tentatively scheduled for Plattsburg. An open house invitation has been extended for inspection of the new Cabot wollastonite processing mill.

A. B. Moore, General Electric Co., spoke to the group on *The Industrial Power System—How to Get the Most for Your Dollar.*

Utah Section members joined with student members at the University of Utah for the annual Joint Dinner Meeting. More than 100 members and guests attended the affair.

Personals

M. D. Banghart is now managing director of O'okiep Copper Co. Ltd. **J. M. MacLeod** is general manager and **E. S. W. Hunt** is manager. Mr. Banghart has also been made managing director of Tsumeb Corp. Ltd. **Charles E. Stott** is general manager. **J. B. Ward** is manager and **J. P. Ratledge** is assistant manager. The anouncement of the changes in both these African firms was made from the New York office of **H. DeWitt Smith.**

Howard A. Meyerhoff is now president of the Scientific Manpower Commission with offices at 1530 P. Street, N.W., Washington, D. C.

Robert L. Hamilton has left Butte, Mont., and is in Chuquicamata as junior geologist with the Chile Exploration Co.

Frank F. Kolbe, president, United Electric Coal Co., Chicago, has been elected to the board of directors of Clark Equipment Co.

Ernest N. Patty and **Merle Tuve** were awarded doctor's degrees at the 1953 commencement of the University of Alaska. Mr. Patty, president of Alluvial Golds Inc., was one of the first six faculty members when the college opened in 1922.

H. I. Altshuler after more than 30 years of foreign mining principally in Latin America is returning to the U. S. Formerly general manager of Frontino Gold Mines Ltd. in Colombia, Mr. Altshuler will make his home at 18 Oak Street, New Canaan, Conn.

Ross L. Dobbin was elected president of The Engineering Institute of Canada in May. Immediately following his induction as president, Mr. Dobbin flew from Montreal to London to represent the Institute in Westminster Abbey for the coronation of Queen Elizabeth II.

Richard V. Colligan, mining geologist, has been made Cuban representative of Freeport Sulphur Co. and its subsidiary, Nicaro Nickel Co. Mr. Colligan is in charge of development work at Moa Bay.

James A. Ross, after 30 years as head of Sprague & Henwood, Scranton, Pa., is now chairman of the board. Succeeding him as president is **Adrian E. Ross. William J. Schank** has been made senior vice president and general manager.

John W. Lampert is now employed by the Aluminum Co. of America at Rosiclare, Ill. as a mining engineer. He was with Resurrection Mining Co., Leadville, Colo.

Horace M. Albright was chosen Alumnus of the Year for 1952 by the alumni of the University of California. Mr. Albright, who graduated in 1912, is president of the U. S. Potash Co., New York, and also widely known for his outstanding work on conservation.

Richard C. Diehl, president of Chase Brass & Copper Co., Waterbury, Conn., was elected president of the Copper & Brass Research Assn., New York, at the annual spring meeting of the Association held recently in Hot Springs, Va.

Joseph F. Harrington, U. S. Geological Survey, is in South America on a Point Four assignment for mineral resources development in Colombia and Ecuador.

C. A. Weekley is in charge of the Ore Dressing Department of the recently announced Soriano y Cia. operations in the Philippines called the Atlas Mining & Development Co. Mr. Weekley was with Marsman & Co. in Manila. He was on the board of directors and chief metallurgist for four mines managed by Marsman.

John F. Haynes of the U. S. Bureau of Mines is now in Tuscon, Ariz., as a field engineer with the Mining Methods Branch of Region IV. He was formerly in Salt Lake City.

Harold L. Childress was named to succeed **O. W. Bilharz** as president of Tri-State Zinc & Lead Ore Producers Association. The offices for the association which covers the Missouri-Kansas-Oklahoma district will continue to be located in Baxter Springs, Kans.

H. W. Straley, III, after spending the field season in geophysical investigations of chromite deposits in North Carolina, is in Princeton, W. Va.

Malcolm B. Gould is now president of H. W. Gould & Co., San Francisco, as well as head of Klau Mine Inc. Mr. Gould plans an intensive exploration program directed toward the production of both metallic and nonmetallic ores.

Elwood S. Moore, retired professor of geological sciences, University of Toronto, has been appointed an associate of Behre Dolbear & Co. with offices in Toronto.

Arnold Ziffzer has left McNally-Pittsburg Mfg. Corp. and is now a consulting mining engineer and geologist on Hiway 62 East, Eureka Springs, Ark.

Franz L. Mayer has left his position as research metallurgist with Patino Mines in Bolivia and is working as research engineer with the Tin Processing Corp., Texas City, Texas.

Roger L. Wensley has accepted chairmanship of the committee of Columbia Engineering School alumni who are conducting a drive to obtain financial support from corporations for Columbia University's new Engineering Center. Mr. Wensley, a 1916 graduate of the School of Mines, is chairman of the board of G. M. Basford Co., an industrial advertising agency.

Lester R. MacLeod, after three and a half years in Chuquicamata with Chile Exploration Co., has returned to the United States. He is in the Engineering Dept. of Anaconda Copper Mining Corp., with his headquarters in New York.

Elton A. Youngberg, recently geologist with the Juneau Office of the U. S. Geological Survey in Alaska, is in Washington, D. C. with the Division of Raw Materials of the Atomic Energy Commission.

Charles Will Wright, mining consultant, has just returned to Washington, D. C. from Mexico where he conferred with the Ministry of Economy and other officials.

WALTER A. STERLING

Walter A. Sterling was elected president and a member of the board of directors of The Cleveland-Cliffs Iron Co., Cleveland, Ohio. Mr. Sterling was formerly vice president in charge of mine operations.

H. D. (Hank) Clark and **Charles W. Yetter** have organized a new group known as Engineers Associates Inc. in Grand Junction, Colo. This organization renders consulting engineering and management services with particular reference to the Colorado Plateau uranium industry.

Francis B. Speaker has returned to Hewitt-Robins, Stamford, Conn. Since the outbreak of the Korean War, he has been in Washington, D. C. as director of the Mining Requirements Division of DMPA.

Clyde W. Hall, who was president of United Clay Mines Corp., Trenton, N. J., is now chairman of the board. He is living in De Land, Fla.

A. Thunaes is now director of research and development for Eldorado Mining & Refining Co. with headquarters in Ottawa, Ont.

A. F. Franz, president of the Colorado Fuel & Iron Corp., received the honorary degree of Doctor of Commercial Science at the 93rd commencement of St. Bonaventure University, New York.

T. L. B. Jepsen is in Trujillo, Peru, as mill superintendent for the Northern Peru Mining & Smelting Co. His former assignment was in Bolivia with Cia. American Smelting Boliviana.

RALPH K. GOTTSHALL

Ralph K. Gottshall was elected president of Atlas Powder Co., Wilmington, Del. Mr. Gottshall, formerly executive vice president, joined Atlas Powder Co. as a chemist in 1927 following his graduation from Lafayette College.

Darwin E. Neihaus received a watch from the Old Timers Club on the recommendation of the faculty as the outstanding senior in coal mining engineering at the University of Illinois. The presentation was made by **Harry M. Moses.** Mr. Neihaus is employed by the Sahara Coal Company, Harrisburg, Ill.

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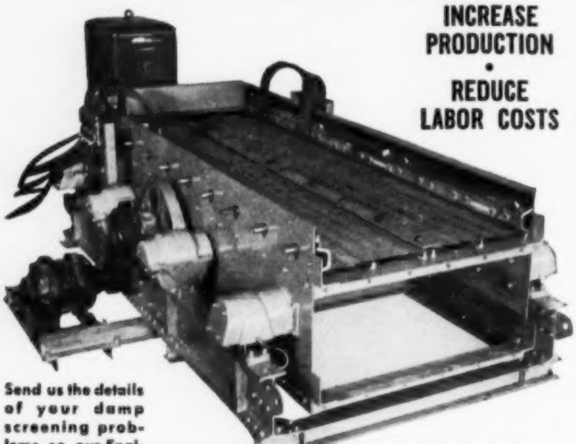

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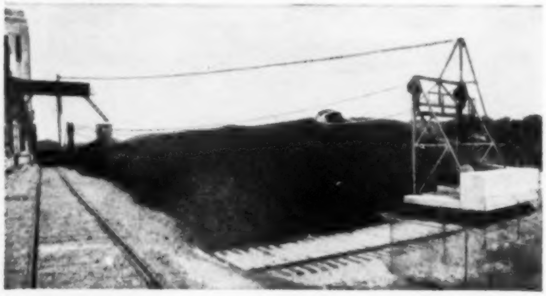

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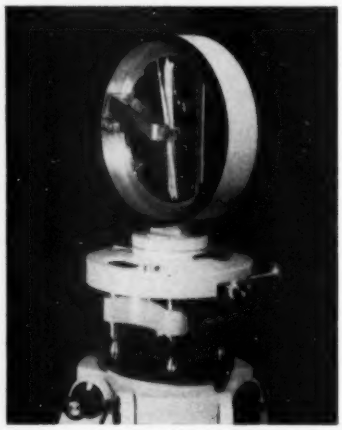

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Obituaries

Memorial Resolution in honor of HOWARD N. EAVENSON

WHEREAS, with the passing of Howard N. Eavenson in Pittsburgh, Pennsylvania, on February 16, 1953, the Institute has lost one of its most loyal and respected members; and

WHEREAS, through his fifty-two years of membership he gave freely of his time in advancing the objectives and elevating the aims of the Institute; and

WHEREAS, he served the Institute as its President in 1934, Vice-President in 1931 to 1933, and Director in 1917-1919, 1930 and 1935-1936, as well as the first chairman and principal organizer of the Institute's Coal Division; and

WHEREAS, he received the Saunders Medal in 1950 and became a Senior Member in 1950 on completion of fifty years' membership in the Institute; and

WHEREAS, he served on many of the Institute's important committees with unusual judgment and distinction, always interested in the advancement of his profession;

THEREFORE BE IT RESOLVED, The American Institute of Mining and Metallurgical Engineers records with a great sense of loss and sorrow the passing of this marvelous friend and highly skilled engineer; and

BE IT FURTHER RESOLVED, that this Resolution be spread on the minutes of this meeting and copy sent to the family of the late Howard N. Eavenson.

By
Newell G. Alford

Hugh Robert Edwards (Member 1916) died Feb. 23, 1953. Mr. Edwards was a consulting engineer in Los Angeles. He was born in Illinois and graduated from the University of Nebraska in 1904. After working as a transitman with a Mexican railroad and as a bridge engineer with Western Pacific Railway, he became chief engineer for Hammon Engineering Co. in California and general manager of Santa Paula Oil Association in San Francisco. He was at one time chief engineer for Anglo-Oriental Min. Corp. in England.

Gerald B. Gould (Member 1934) died Mar. 20, 1953. Mr. Gould had been president of the Fuel Engineering Co. of New York since 1928 and a partner since 1938 in the New York firm of Gould, Henderson & Brunjes. During World War II Mr. Gould served as an advisory consultant on coal prices in charge of the solid fuel section of the OPA. He was born in Detroit and received his A.B. from Yale in 1910. After receiving his M.A. from the University of Pennsylvania in 1911, Mr. Gould joined the Fuel Engineering Co. He was the author of several books on his speciality, including "Judging Coal Values" and "Steam Generation Steps Ahead." He also contributed pamphlets for conventions of the American Society for Testing Materials and was a member of AIME's Papers and Publications Committee for a number of years.

Appreciation of **Maurice William Summerhayes** by R. L. Healy

Maurice William Summerhayes, formerly managing director of Wright-Hargreaves Mines Ltd. and at the time of his death consulting engineer of that company, died in Duncan, B. C. on April 10th last, at the age of 81 years.

He was born in England and was a graduate of the Royal School of Mines. His early years as a mining engineer took him to various parts of South America, Mexico and southwestern United States. He spent the longest period of his professional life in northern Ontario, as operator and consulting engineer in Cobalt and in the Kirkland Lake and Porcupine Camps. For several years he was manager of the Porcupine Crown Reserve Mine in Timmins in the early life of that company. He was manager of Wright-Hargreaves Mine from 1929 to 1949 until he retired from its active management in that year.

Mr. Summerhayes was a member of the Canadian Institute of Mining & Metallurgy, the AIME, the Institute of Mining & Metallurgy and a former president of the Ontario Mining Association. He took a keen interest in sailing, hunting and fishing and in his youth was a noted sprinter.

He was ever a kindly and modest gentleman, with a host of friends who will sympathize with his two daughters, Mrs. Dorothy Barclay and Phyllis Summerhayes of Duncan, B. C., in his passing.

NECROLOGY

Date Elected	Name	Date of Death
1904	Clarence E. Abbott	Apr. 14, 1953
1951	Harry Dobrin	October 1952
1898	Harlan S. Emlaw	Feb. 5, 1953
1922	Elmer H. Finch	Apr. 29, 1953
1920	R. E. Gruber	January 1953
1939	Willard M. Hall	Apr. 9, 1953
1942	B. C. Leadbetter	Oct. 10, 1952
1916	Charles Lindmueller	May 3, 1953
1931	Sture Mortsell	1951
1931	Francis M. Walters, Jr.	Apr. 18, 1953

Proposed for Membership

MINING BRANCH, AIME

Total AIME membership on May 31, 1953 was 18,822; in addition 1227 Student Associates were enrolled.

ADMISSIONS COMMITTEE

O. B. J. Fraser, Chairman; Philip D. Wilson, Vice-Chairman; F. A. Ayer, A. C. Brinker, R. H. Dickson, Max Gensamer, Ivan A. Given, Fred W. Hanson, T. D. Jones, George N. Lutjen, E. A. Prentis, Sidney Rolle, John T. Sherman, Frank T. Sisco, R. L. Ziegfeld.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

Alabama
Bessemer—Boyle, James R. (R. C/S—S-J)

Arizona
Ajo—Jones, James S. (M)

California
Altadena—Pray, Lloyd C. (C/S—A-M)
Menlo Park—Knight, Joseph F. (A)
Nevada City—Eastman, Byron L. (R. M)

Idaho
Kellogg—Jasberg, Onni J. (R. M)
Rathdrum—Golding, Charles L. (R. C/S—A-J)

Michigan
Birmingham—Bodor, Robert L. (M)
Caspian—Pirolo, Dino H. (J)

Missouri
Kansas City—Banghart, Roger C. (J)

Nevada
Henderson—Love, Frank E. (R. M)
Reno—Wingfield, George (R. C/S—A-M)
Reno—Isbell, Clement V. (M)

New Mexico
Santa Rita—Leroy, Paul G. (J)
Vanadium—McClain, Charles W. (J)

New York
Bayside—Persons, Robert W. (M)
Lewiston—Barnes, John F. (R. C/S—J-M)
New York—Macomber, W. Sturgis (A)
Plandome—Zeckhausen, Paul W. (A)

Ohio
Powell—Mangio, Charles A. (J)

Oregon
Grants Pass—Robertson, William S. (M)
Portland—Meyers, Joseph D. (J)

Pennsylvania
Johnstown—Kodrowsky, William G. (R. C/S—S-J)
Tamaqua—Smith, James D. (R. C/S—S-M)

Texas
Dallas—Dallas, Orlan L. (M)

Utah
Salt Lake City—Whisenant, Robert N. (M)

Virginia
Falls Church—Keller, George V. (J)

Wyoming
Rock Springs—Douglas, Sanford J. (M)

Africa
British West Africa, N. Nigeria—Blazewicz, Kazimierz (M)
S.W. Africa, Grootfontein—Halupka, Herman A. (M)
Union of South Africa, O'okiep, C.P.—MacLeod, John M. (M)

Australia
Roseberry, Tasmania—Henkel, John S. (R C/S—S-J)

Canada
Ontario, South Porcupine—Girdwood, Charles P. (M)

China
Singapore—Tillia, Henry J., Jr. (R. M)

Cuba
Santiago—Soto Bucarely, Luis B. (A)

Mexico
Durango, Dgo.—Maldonado, Carlos F. (M)

Norway
Sauda—Reinertsen, Kristen (M)

Peru
Cerro de Pasco—Iten, Ken W. B. (M)
Lima—Bell, Jerry Y. (M)
Lima—DeSzepessy Schaurek, Ali (J)
Trujillo—Vanags, John F., Jr. (M)

Philippines
Mt. Province—Franco, Evaristo B. (M)

Geophysicists
Drilling
Management
Metallurgical
Reports
Valuations

—Coming Events—

July 13-16, Golden Jubilee of the Idaho Mining Assn., Sun Valley, Idaho.

Aug. 31-Sept. 2, Minerals Beneficiation Division, Hibbing, Minn.

Sept. 8-13, Joint Meeting Industrial Minerals Division, AIME, Keltic Lodge, Ingonish, Nova Scotia.

Sept. 11, AIME, St. Louis Lead Belt Meeting, Tour of St. Joseph Mines and Mills, St. Louis.

Sept. 13-16, Electrochemical Chemical Society Inc., Ocean Terrace Hotel, Wrightsville Beach, N. C.

Sept. 21, ASME, Industrial Instruments and Regulators Div., and **Instrument Society of America**, exhibit and joint conference, Sherman Hotel, Chicago.

Sept. 21-24, American Mining Congress Mineral Mining Convention, Olympic Hotel, Seattle.

Oct. 1-2, AIME, Regional Meeting, Pacific Petroleum Chapter, Ambassador Hotel, Los Angeles.

Oct. 6, AIME, National Open Hearth Steel Committee, Buffalo Section.

Oct. 7, AIME, National Open Hearth Steel Committee, Chicago Section, Phil Smidt's.

Oct. 7-9, National Assn. of Consulting Engineers, south central region, Mayo Hotel, Tulsa, Okla.

Oct. 8-9, Ninth National Conference on Industrial Hydraulics, Hotel Sheraton, Chicago.

Oct. 9, AIME, St. Louis Local Section, St. Louis University, St. Louis.

Oct. 9, AIME, National Open Hearth Steel Committee, Eastern Section, Fall Meeting, Warwick Hotel, Philadelphia.

Oct. 15-17, AIME, Second Annual Clay Minerals Meeting, University of Missouri, Columbia, Mo.

Oct. 15-17, AIME National Open Hearth Steel Committee, Southwestern Section, Fall Meeting, Baker Hotel, Dallas.

Oct. 18-21, AIME, Petroleum Branch Fall Meeting, Baker and Adolphus Hotels, Dallas.

Oct. 19-21, AIME, Institute of Metals Division, Fall Meeting, Hotel Allerton, Cleveland.

Oct. 19-23, Forty-first National Safety Congress and Exposition. Sessions on industrial safety, Conrad Hilton, Congress, Morrison, and Hamilton Hotels, Chicago.

Oct. 22, AIME, St. Louis Local Section, Engineers Club of St. Louis and ASCE, Engineers Club, St. Louis.

Oct. 27, Assn. of Consulting Chemists & Chemical Engineers, 25th Anniversary Annual Meeting, Hotel Belmont Plaza, New York.

Oct. 28-31, AIME, El Paso Meeting, in cooperation with International Mining Days, Hotel Paso Del Norte, El Paso.

Oct. 29-30, AIME, ASME Fuels Conference, Conrad Hilton Hotel, Chicago.

Oct. 29-31, Annual Meeting of National Council of State Board of Engineering Examiners, Plaza Hotel, San Antonio.

Oct. 30-31, AIME, National Open Hearth Steel Committee, Deshler-Wallick, Columbus.

Nov. 6, AIME, National Open Hearth Steel Committee, Pittsburgh Section, William Penn Hotel, Pittsburgh.

Nov. 13, AIME, St. Louis Section, Coal Meeting, York Hotel, St. Louis.

Nov. 29-Dec. 4, ASME, Annual Meeting, Statler Hotel, New York.

Dec. 2-4, Electric Furnace Steel Conference, Netherland Plaza Hotel, Cincinnati.

Dec. 11, AIME, St. Louis Section, York Hotel, St. Louis.

Dec. 13-16, American Institute of Chemical Engineers, Annual Meeting, Hotel Jefferson, St. Louis.

Dec. 28-29, Annual Chemical Engineering Symposium, University of Michigan, Ann Arbor.

Advertisers' Index

* Previous Issues

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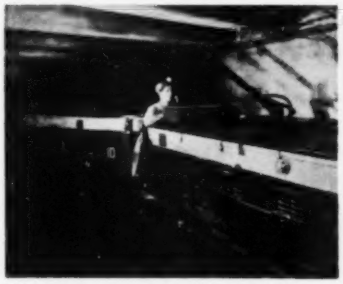
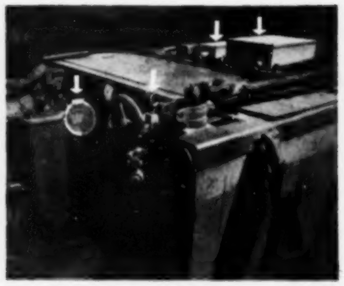

Motorman reports his position at cross-over.


Motorman, ready to go into mine, asks for right of way.


Trip unloader advises conditions at his station.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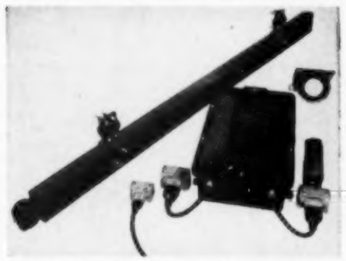
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